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New Mine Sealing Techniques For Water Pollution Abatement

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27 **Abstract**

The purpose ~~of this project~~ was to develop and field test new concepts for watertight mine seal and bulkhead construction applicable to abatement of acid mine water pollution. Laboratory research determined proper materials, equipment and techniques for constructing mine seals or bulkheads. Field testing was conducted on remedial grouting techniques for constructing mine seals or bulkheads. Two new processes were developed. One involved a technique of placing a plug of graded limestone aggregate in a mine drift or portal to neutralize an acid mine water discharge until a seal was effected. In the process, precipitation of iron hydroxide gradually closed the pores. A pneumatic conveying technique permitted placement of up to 550 pounds per minute of aggregate into a mine drift. The second process consisted of remotely constructing a mine seal including rear and front bulkheads of a self-supporting, quick-setting sodium silicate cement specifically developed for this application. A filler material of expansive cement was used between the bulkheads to complete the seal. Field testing in West Virginia substantiated the feasibility of both processes when two aggregate and two bulkhead type seals were placed in abandoned mines which had drainage flows up to 58 gallons per minute. Cost of the seals are reported.

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NEW MINE SEALING TECHNIQUES FOR WATER POLLUTION ABATEMENT

by

Halliburton Company
Duncan, Oklahoma 73553

for the

FEDERAL WATER QUALITY ADMINISTRATION

DEPARTMENT OF THE INTERIOR

Program Number 14010 DMO

FWPCA Contract No. 14-12-453

March, 1970

WATER POLLUTION CONTROL RESEARCH SERIES

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ABSTRACT

The purpose of this project was to develop and field test new concepts for watertight mine seal and bulkhead construction applicable to abatement of acid mine water pollution. Laboratory research determined proper materials, equipment and techniques for constructing mine seals or bulkheads. Field testing was conducted on remedial grouting techniques for constructing mine seals or bulkheads. Two new processes were developed. One involved a technique of placing a plug of graded limestone aggregate in a mine drift or portal to neutralize an acid mine water discharge until a seal was effected. In the process, precipitation of iron hydroxide gradually closed the pores. A pneumatic conveying technique permitted placement of up to 550 pounds per minute of aggregate into a mine drift. The second process consisted of remotely constructing a mine seal including rear and front bulkheads of a self-supporting, quick-setting sodium silicate cement specifically developed for this application. A filler material of expansive cement was used between the bulkheads to complete the seal. Field testing in West Virginia substantiated the feasibility of both processes when two aggregate and two bulkhead type seals were placed in abandoned mines which had drainage flows up to 58 gallons per minute. Cost of the seals are reported. This report was submitted in partial fulfillment of Contract No. 14-12-453 between the Federal Water Pollution Control Administration and the Halliburton Company.

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INTRODUCTION

Synopsis of Previous Study

A previous contract to the Halliburton Company consisted of a feasibility study in four parts on the application of various grouting agents, techniques and methods to the abatement of mine drainage pollution. This study, conducted from December 1966 to May 1968, included a survey of mine drainage pollution in the Upper West Fork River Sub-basin near Clarksburg, West Virginia. This area is located on the map attached as Figure 1.

The Monongahela River is formed by the confluence of the West Fork and Tygart Valley Rivers, which occurs approximately five miles north of the northern boundary of Harrison County, near Fairmont, West Virginia. The Halliburton study was concerned primarily with that portion of the West Fork River Basin which terminates immediately south of Clarksburg in Harrison County and extends to the headwaters in Upshur County. The Sub-basin contains approximately 400 square miles and more than 200 drift mines.

Clarksburg, the fifth largest city in the State and seat of Harrison County, is the major center of population and industry in the Upper West Fork River Sub-basin. Situated at the base of the Sub-basin, the city obtains its water supply directly from the West Fork River. Numerous other communities are interspersed throughout the Sub-basin and also take their water supplies from the river or its tributaries.

The feasibility study included an initial survey made by reconnaissance parties throughout the Upper West Fork Sub-basin. All mined areas as revealed from information furnished by the Monongahela River Mine Drainage Remedial Project were included in the survey.

The reconnaissance party located and identified the openings which were evident within the Sub-basin and drainage water was sampled and analyzed to determine the acid and mineral content. The geology of the mine sites was also recorded. A total of 228 drift mines or openings were discovered in this survey. A resurvey was then conducted on 60 mines which evidenced measurable flows during the preliminary survey, or had drainage with high mineral content.

Possible remedial techniques to abate the flow of mine drainage from the mines in this area were studied and a listing of the various methods was made. A drift mine which had an acidic drainage coming from the mine opening was selected for the installation of a mine seal. The seal installed consisted of four layers of cloth retainers which were progressively filled with a quick-setting cement grout. Each layer was allowed to conform to the shape of the mine floor, walls or roof as the material hardened. When complete, the stack of retainers formed a seal which conformed to the shape of the opening.

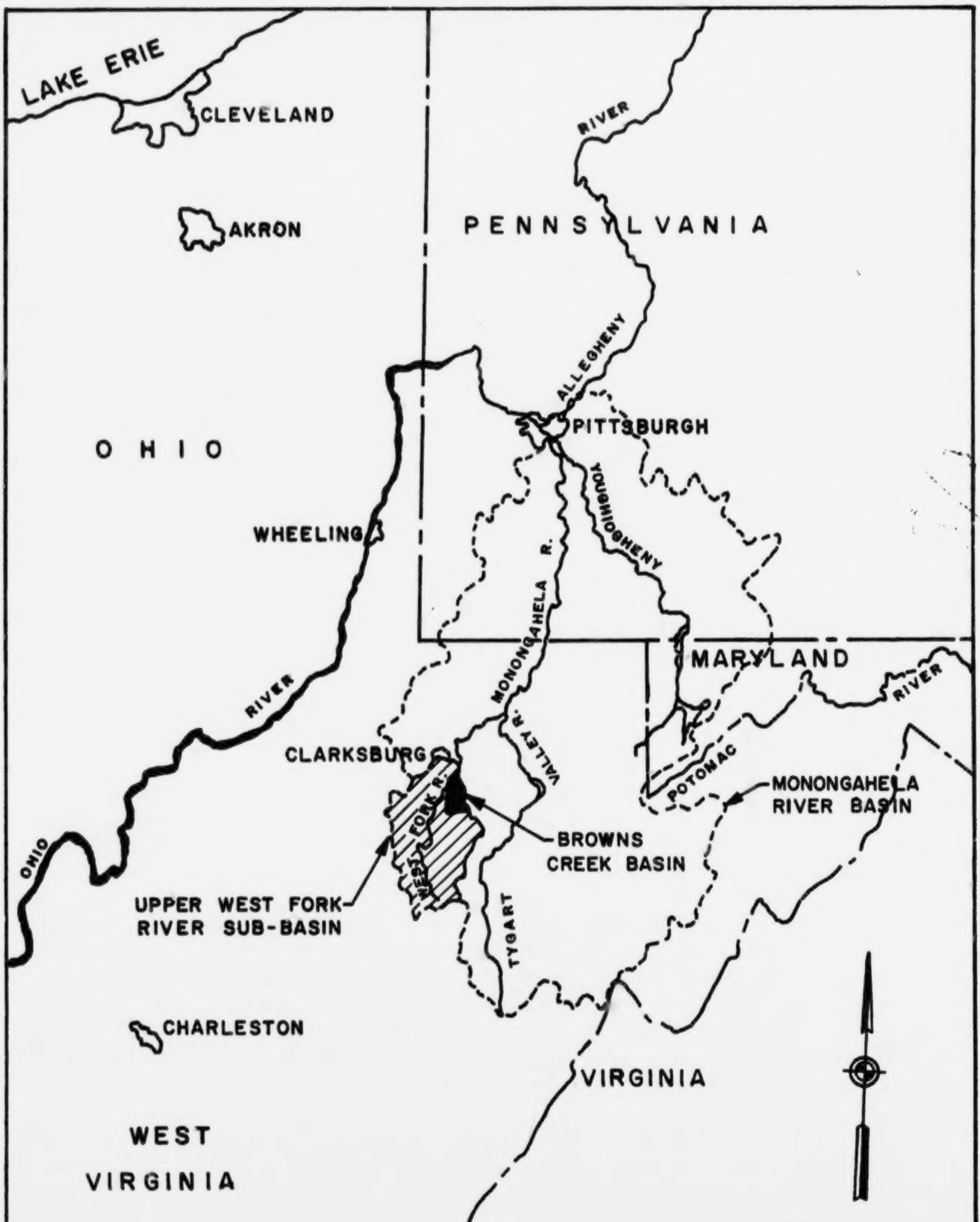
LAKE ERIE
CLEVELAND
AKRON
PENNSYLVANIA
RIVER
ALLEGHENY
OHIO
PITTSBURGH
YOUGHIOGHENY
WHEELING
MONONGAHELA R.
MARYLAND
RIVER
RIVER
CLARKSBURG
VALLEY R.
WEST FORK R.
POTOMAC
MONONGAHELA RIVER BASIN
BROWNS CREEK BASIN
OHIO
UPPER WEST FORK RIVER SUB-BASIN
TYGART
CHARLESTON
VIRGINIA
WEST VIRGINIA
0
25
50
STATUTE MILES

FIGURE I - GENERAL LOCATION MAP

Three other mine complexes were selected which were felt to be representative of the flow and acid drainage of those within the Basin. Complete plans and specifications were drawn up for abatement construction work on the mine openings, utilizing the grout retainer seal technique as well as other sealing methods.

Additional work in the study included research on the grout retainer technique of sealing a mine, with emphasis being given to development of a cement slurry which would permit the seal to be placed in one working day. This would reduce the cost which had been incurred on the one already placed, since that application took several days. A similar type retainer was developed for use in 40-inch auger holes. Several seals were successfully placed in auger holes in field trials. Two mine drifts were sealed remotely by the technique of placing aggregate in the mine pneumatically and subsequently grouting this aggregate to form a 25-foot plug in each mine drift opening.

Scope and Purpose of Present Study

The work reported herein is a sequel to the previous feasibility study. The first phase of the work involved remedial procedures to correct leakage which was issuing from the mines sealed in the previous contract. The seal installed in Mine No. 14-042A was not leaking, but there was water issuing from a subsided area at the left front of the mine seal at a rate of about one-half gallon per minute. The two openings of Mine No. 40-016 had previously been sealed by placing aggregate plugs in the mine and grouting the aggregate. Water was leaking from these openings at the rate of about 3 gallons per minute from Opening No. 1 and 4.5 gallons per minute from Opening No. 2.

Another phase of the present study was a research investigation to improve the pneumatic method for placement of aggregate into a mine drift and thereby make the process more economically feasible.

Additional research was conducted to determine other techniques of seal installation and construction which might provide an economic or practical advantage over the grout retainers used in the mine seal constructed in the previous study. The unsuitability of the grout retainer seal in mine drifts containing center posts was another consideration for further research in mine seals.

Research was also conducted to determine the feasibility of placing a permeable plug within a mine drift. A permeable plug is a partial seal which allows the escape of some mine water through the plug and at the same time provides in-place treatment for the fluid as it passes through the plug.

A high-flow mine was selected for the field application of the techniques developed in the other phases.

CONCLUSIONS

Numerous findings have resulted from the tests and evaluations performed under this contract. The following conclusions are drawn from these findings:

Bulkhead Development

(1) A system was successfully developed to move aggregate pneumatically at rates up to 550 pounds per minute with a minimum amount of downtime. The method utilized two alternately loaded and discharged pneumatic tanks. This could approach a continuous operation by increasing the size of the tanks and/or the number of tanks.

(2) It was found that pneumatic handling of aggregate which contains fine or powdered limestone and any appreciable amount of moisture tends to clog the conveying pipe. Every effort should be made to keep the material dry while loading or conveying.

(3) To convey aggregate pneumatically from pressurized tanks, the use of a special air manifold as described in the report is mandatory.

(4) Investigation of two organic foam materials led to the conclusion that neither would be practical for building bulkheads in drift mine applications. Equipment and material costs, together with knowledge of uncontrollable mine atmospheric conditions which could adversely affect foam density, were the basis for elimination of polyurethane foam. Urea-formaldehyde foam was concluded as unsatisfactory because of low strength, low bulk density, high equipment costs and poor resistance to biological, dilute acid or water attack.

(5) Laboratory investigation of inorganic foam materials for possible use in building temporary drift mine bulkheads indicated that these materials were fragile with poor water resistance. It was concluded that these characteristics would make this material unsatisfactory for bulkheads in mine drifts.

(6) A sodium silicate cement slurry with quick-setting and self-supporting properties was developed. This material was satisfactory for use in remotely building bulkheads in mine drifts.

(7) Special equipment for mixing and placing the quick-setting slurry was developed. Techniques using this equipment and the special slurry were proven to be satisfactory. It is concluded that this method is a practical and applicable method for construction of mine seals.

(8) Based on the results of field tests, it was concluded that a mine seal, composed of rear and front bulkheads of a quick-setting slurry with expansive type cement filler between, would be satisfactory to seal a mine drift to hold a required head of impounded mine water.

(9) Obtaining access rights for field applications is difficult and can require a considerable expenditure of time and money.

Permeable Plug Investigation

(10) Laboratory investigation determined that a graded limestone aggregate could be used as a plug to neutralize acid mine water passing through the plug as it gradually seals the pore space due to precipitation of iron hydroxide. The initial success on a field application of a plug of graded aggregate in a mine drift tends to confirm this laboratory finding. Based on these findings, this method appears promising as a practical approach to acid mine water impoundment and in-place treatment.

Sealing of a High Flow Mine

(11) Field tests were conducted to determine the feasibility of placing a mine seal in a high flow mine. Based on findings from these tests, it is concluded that a mine seal can be constructed in a high flow mine using the techniques developed under this contract.

(12) It was concluded that an expansive cement filler is preferred to a grouted aggregate filler for double bulkhead type mine seals when experience to date in constructing mine seals, ease of placement, economic factors and possible need of remedial work are considered.

(13) The use of agricultural lime blended in the graded aggregate caused dust problems. The use of a water spray at the discharge helped some but did not solve the problem. It was concluded that this problem must be solved before powdery material can be used extensively.

Remedial Procedures - Mines No. 14-042A and 40-016

(14) The injection of gelled fluid into a previously sealed mine was not successful in reducing leakage. No evidence of gelled fluid return was observed at the point of leakage on Mine No. 14-042A. It was concluded that the gelled fluid did not communicate with the leakage point. Such communication is essential in order to seal with this type of treatment.

(15) Remedial grouting with cementitious material was conducted unsuccessfully on an aggregate seal which was placed and grouted in Mine No. 40-016 under a previous contract. It is apparent that it is very difficult to consolidate aggregate in place and obtain a satisfactory seal due to the necessity of containing the grout within the aggregate. Although many of the fluid channels were apparently closed during the remedial grouting, enough additional channels remained open to permit the flow to continue unabated. Because of the rate of leakage through these two grouted aggregate seals which were emplaced during the previous contract and the lack of success in sealing the leaks, it is concluded that the grouted aggregate type seal is not practical.

RECOMMENDATIONS

Based on the conclusions drawn in this study, the following recommendations are submitted:

(1) When conditions permit, it is recommended that drift openings of abandoned mines be sealed utilizing rear and front bulkheads of self-supporting cement and a light, expansive type cement between the bulkheads.

(2) It is recommended that methods other than placement of aggregate in drift openings with subsequent grouting be used as a mine seal.

(3) It is recommended that further development work be undertaken to provide more efficient placement equipment for the construction of self-supporting bulkheads.

(4) It is recommended that research be conducted to develop the equipment and techniques for constructing quick-setting bulkheads in inaccessible mine drifts through vertical bore holes.

(5) It is recommended that further research be conducted to determine the optimum materials for use in constructing a permeable plug type mine seal.

(6) It is recommended that further research include a study to determine a suitable means of eliminating the extreme dust problem encountered when pneumatically conveying dusty material into mine openings.

(7) It is recommended that efforts to obtain access rights to property be initiated well in advance of any proposed field work.

REMEDIAL PROCEDURES - MINES NO. 14-042A & 40-016

Two mines, sealed during a previous contract, were selected for remedial work. Mine No. 14-042A had been closed by the use of four expendable grout retainers. Expendable grout retainers were fabricated cloth containers of nylon or cotton, varying in length from 20 feet for the bottom retainer to 10 feet for the top. Each retainer was successively filled with cement grout slurry to conform to the shape of the drift opening. Leakage at the rate of 1/2 gallon per minute was flowing from a fractured coal surface to the left of the seal. The work was performed in an attempt to stop or significantly reduce this leakage.

Two drift openings were exposed at Mine No. 40-016. These openings had been sealed by aggregate plugs 25 feet in length which were placed in the mine pneumatically and grouted. Leakage was observed under the base and across the front width of each plug. Opening No. 1 was flowing about 3.0 gallons per minute and 4.5 gallons per minute was leaking from Opening No. 2. The remedial grouting was performed in an attempt to reduce or stop the leakage.

A map showing the location of these two mines is attached as Figure 2.

Remedial Work - Mine No. 14-042A

This is a small isolated mine of approximately 5 acres located about one mile south of Clarksburg, West Virginia and east of the West Fork River and State Secondary Route 25. The Pittsburgh coal seam has been extensively drift mined and partially strip mined, leaving negligible coal reserves. The coal dips steeply to the east.

A flow of 18 gallons per minute from the mine was measured just prior to sealing the drift opening. The flow discharges into Doll Run, a tributary to the West Fork River. The mine water was found to have a pH of 2.6, an acidity of 2750 mg/l in terms of $CaCO_3$ and an iron content of 558 mg/l. The acid production rate was 616 pounds per day.

Mining in this area began in the early part of the 20th Century. Most of the mines were closed in the 1930's or prior to World War II. An extensive search was made for a map of this mine, but one could not be located. A topographic map was obtained and is included as Figure 3. A geologic section of the mine is included (Figure 63 - Appendix).

This mine was selected for application of a mine seal because of its remote location, small but easily accessible area and highly acidic discharge. The use of expendable grout retainers for a seal installation was selected on the basis of research tests. This type seal consisted of successive layers of expendable grout retainers filled with a cement grout slurry. An expendable grout retainer was a rectangular nylon or cotton cloth container which varied in size from 20 x 10 x 3 feet to 10 x 8 x 3 feet.

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As the seal was installed, the 20-foot long nylon retainer was placed on the floor of the mine and inflated with cement slurry to conform to the shape of the opening. After the first retainer was sufficiently hard to withstand a load of about 3 psi, a second nylon retainer which was 16 x 10 x 3 feet was placed on the first. Figure 4 shows this retainer in place against the walls of the mine. The process was repeated using a third nylon retainer which was 14 x 10 x 3 feet. The 10-foot cotton retainer was used to obtain sufficient height to completely fill and seal the opening. A drawing of this seal showing the installation and mine configuration is attached as Figure 5. It should be noted that the roof and walls of the mine were of coal and the floor was slate. The coal was irregular in shape and contained many fractures, as can be clearly seen in Figure 4. A detailed account of this installation is given in the Halliburton Company report, Part II - "Selection and Recommendation of Twenty Mine Sites", August 23, 1967, p. 137.

After the mine seal was installed, the cumulative leakage from small leaks in the coal surfaces around the seal was measured at 1.5 gallons per minute, a reduction of 92% from the leakage measured from this mine before the seal installation.

Preparation was made to reduce this leakage by grouting around the lower grout retainer using Halliburton PWG grout fluid. This fluid was an acrylamide monomeric solution which had a 10-second set time. An injection of 20 gallons of grout reduced the flow to 0.75 gallons per minute. Further grouting around the second retainer using 30 gallons of grout reduced the leakage to 0.33 gallons per minute. It then appeared that the remaining leakage was coming through coal fractures several feet to the left of the seal face, so grouting was discontinued. Figure 6 shows a view of this mine seal at the time of the present contract initiation.

The plan for remedial work on the present contract was to grout the leakage points with the same grout fluid used previously. Before this work was initiated, the approach was changed. Accordingly, the Statement of Work was modified to conduct remedial measures by injecting a low viscosity polymer or gelling agent into the mine behind the seal to attempt to seal the points of leakage by flowing the material into the leakage points from the inside of the mine.

A gel material made using Karaya gum was considered, but was rejected when it was found that the price of $0.415 per pound for this material would result in a high treatment cost. An alternate material selected was a gel material consisting of bentonite, shredded cane fiber and water. The cost for the dry materials was $0.028 per pound. The cost for the gelled fluid after placement in the mine was $0.069 per gallon.

Prior to beginning remedial operations, an initial measurement was taken of the water level of the impounded water. The measurement

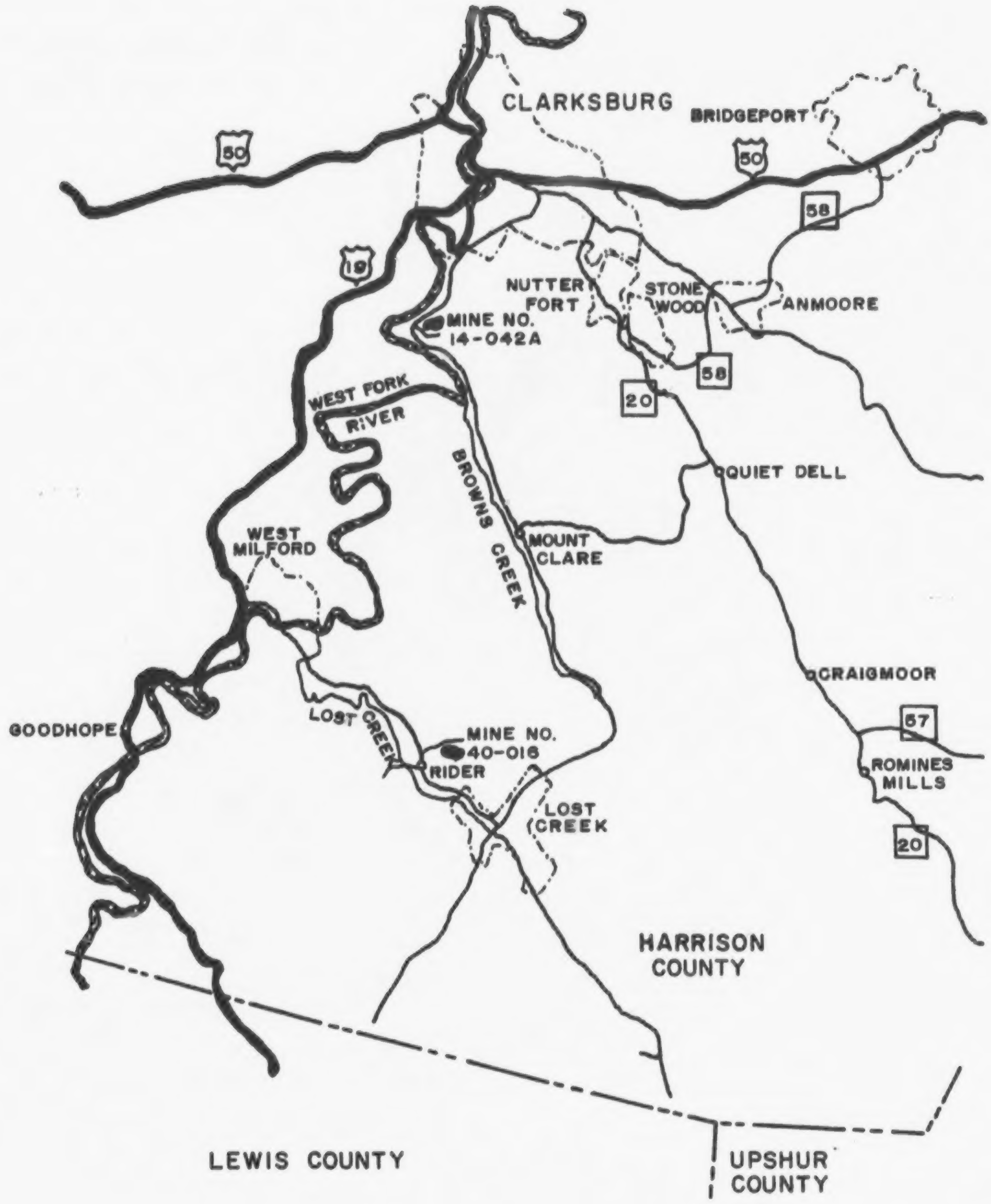

CLARKSBURG
BRIDGEPORT
50
50
58
19
NUTTER FORT
STONE WOOD
ANMOORE
MINE NO. 14-042A
58
20
WEST FORK RIVER
BROWNS CREEK
QUIET DELL
MOUNT CLARE
WEST MILFORD
CRAIGMOOR
57
GOODHOPE
LOST CREEK
MINE NO. 40-016
RIDER
ROMINES MILLS
LOST CREEK
20
HARRISON COUNTY
LEWIS COUNTY
UPSHUR COUNTY

FIGURE 2 - MINE LOCATION MAP

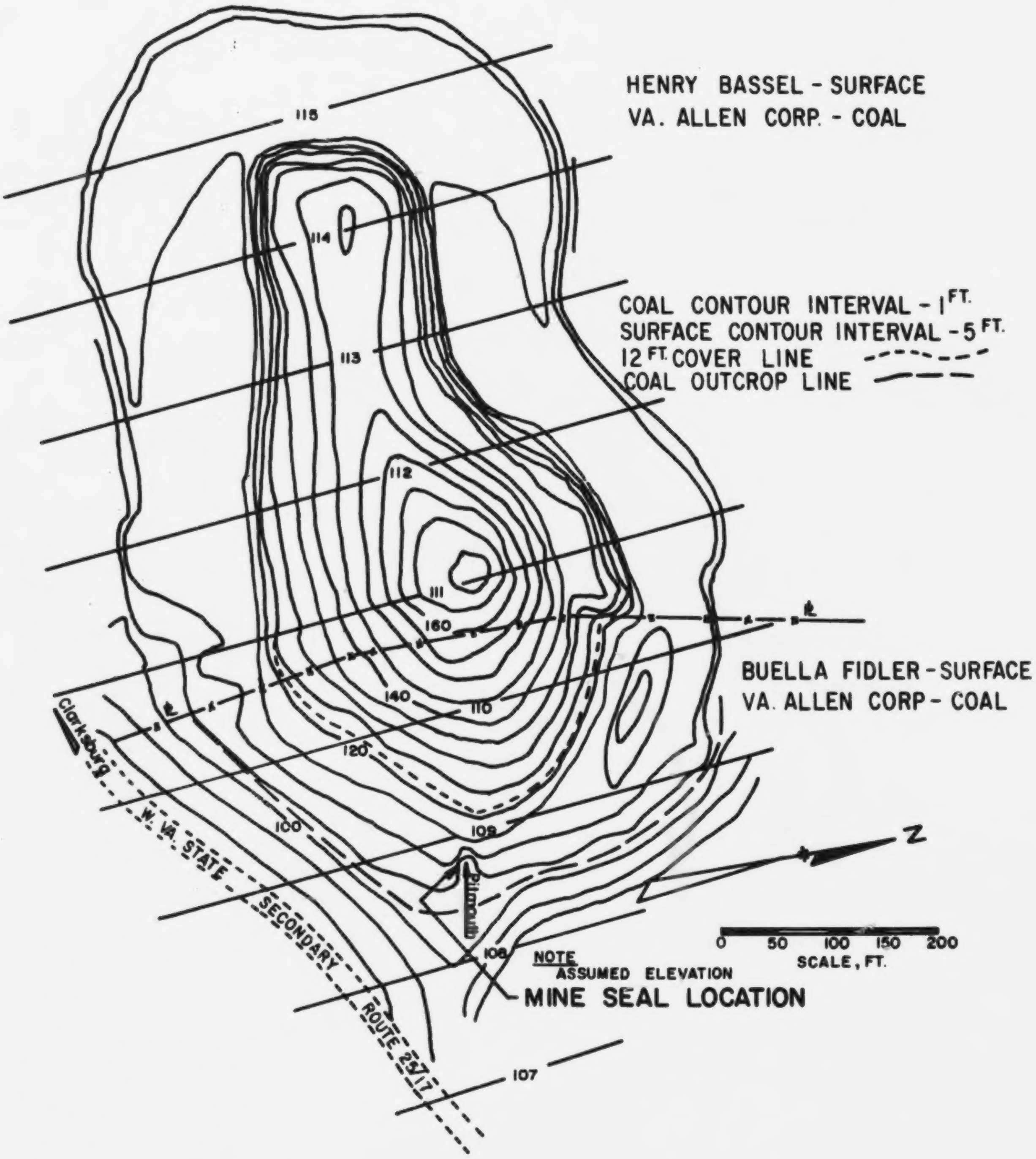

HENRY BASSEL - SURFACE
VA. ALLEN CORP. - COAL
115
114
113
112
111
110
109
108
107
160
140
120
100
COAL CONTOUR INTERVAL - 1 FT.
SURFACE CONTOUR INTERVAL - 5 FT.
12 FT. COVER LINE
COAL OUTCROP LINE
BUELLA FIDLER - SURFACE
VA. ALLEN CORP - COAL
Clarksburg
W. VA. STATE SECONDARY ROUTE 25/17
Pilmouth
N
0 50 100 150 200
SCALE, FT.
NOTE
ASSUMED ELEVATION
MINE SEAL LOCATION

FIGURE 3-TOPOGRAPHIC MAP-MINE NO 14-042A

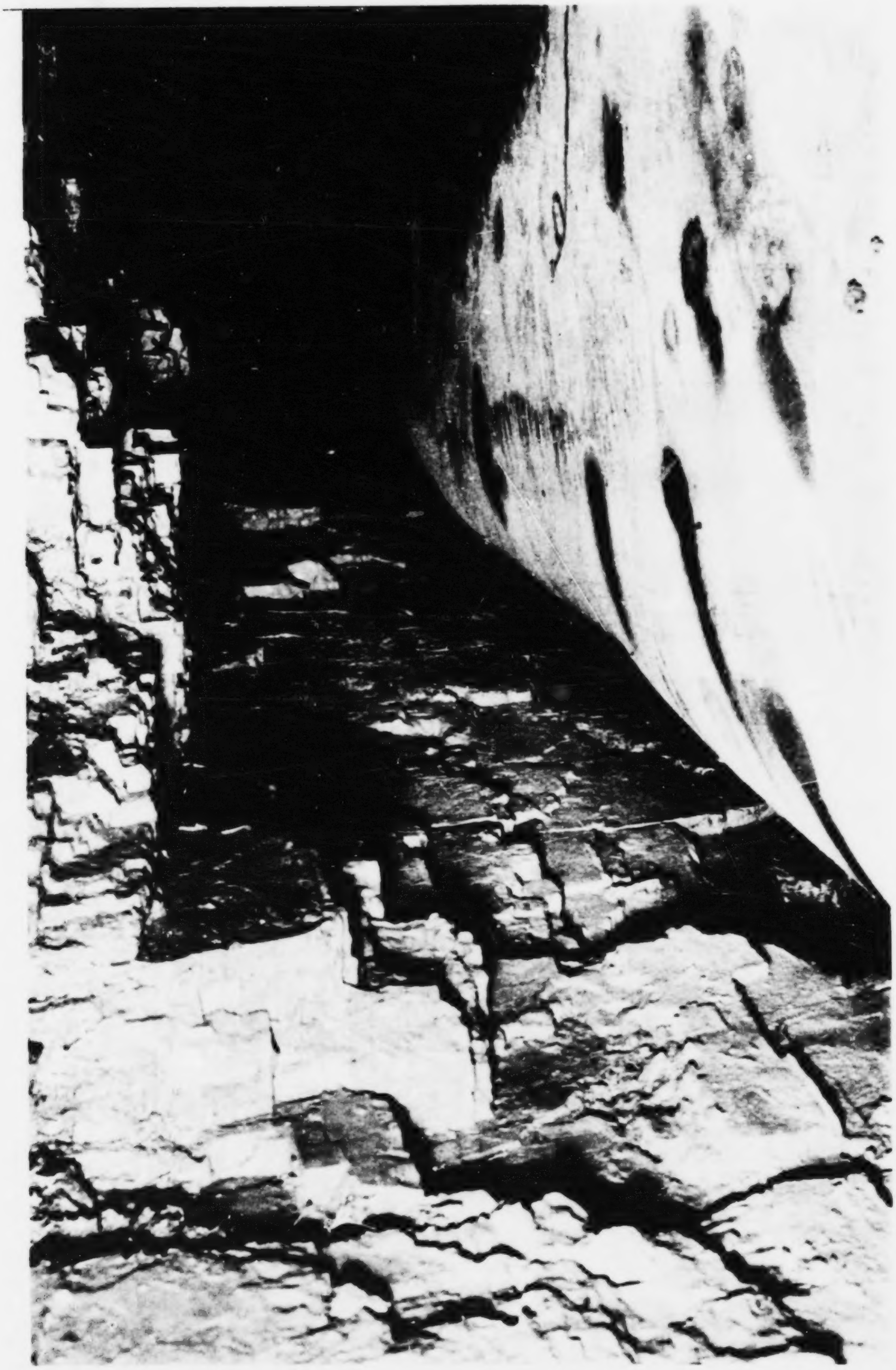
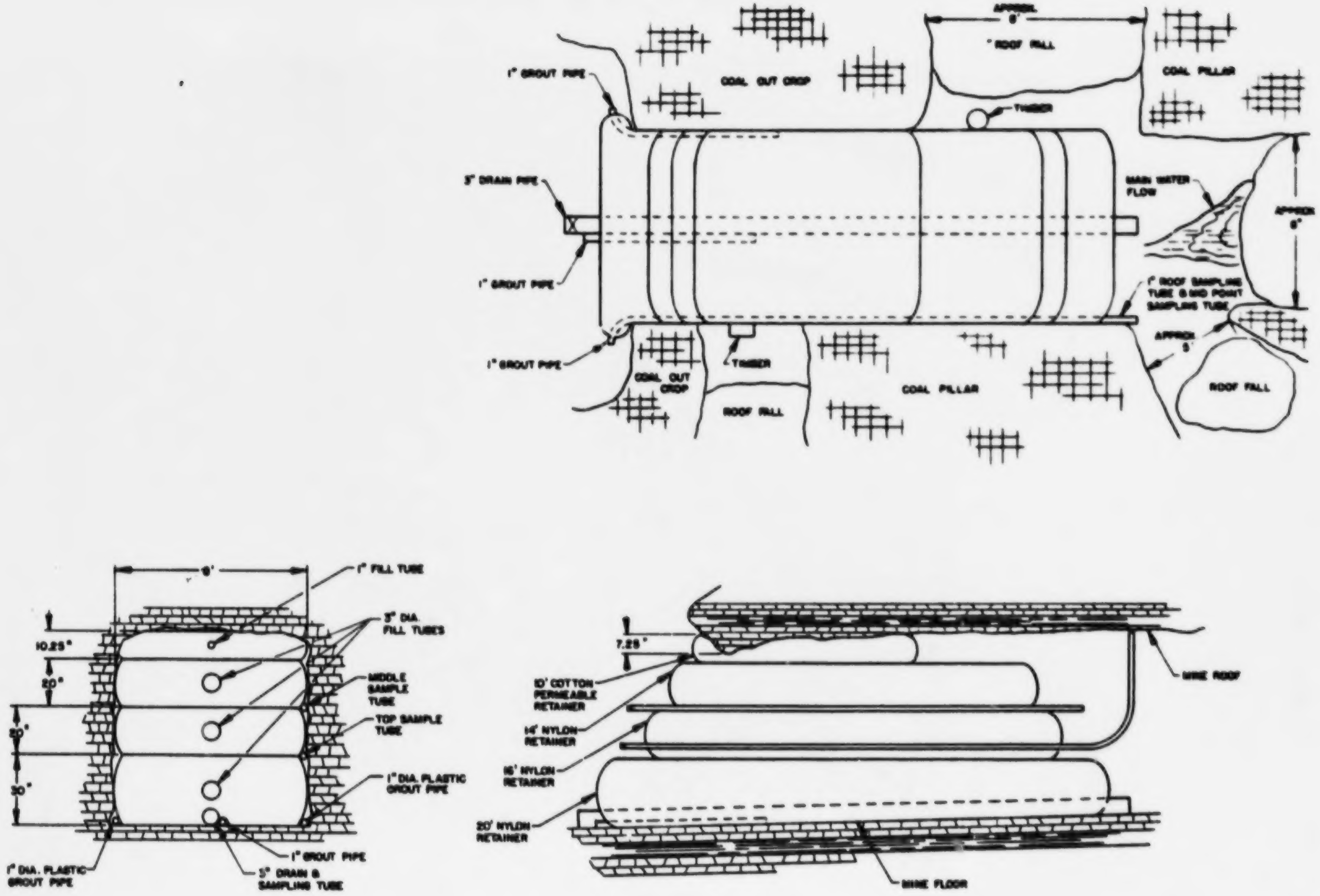


FIGURE 4 - VIEW OF FRACTURED COAL IN WALLS AND ROOF - MINE NO. 14-042A


APPROX. 8'
" ROOF FALL
COAL PILLAR
1" GROUT PIPE
COAL OUT CROP
TIMBER
3" DRAIN PIPE
MAIN WATER FLOW
APPROX. 8'
1" GROUT PIPE
1" ROOF SAMPLING TUBE & MID POINT SAMPLING TUBE
1" GROUT PIPE
COAL OUT CROP
TIMBER
ROOF FALL
COAL PILLAR
APPROX. 5'
ROOF FALL
8'
1" FILL TUBE
10.25"
3" DIA. FILL TUBES
20"
MIDDLE SAMPLE TUBE
20"
TOP SAMPLE TUBE
30"
1" DIA. PLASTIC GROUT PIPE
1" DIA. PLASTIC GROUT PIPE
1" GROUT PIPE
3" DRAIN & SAMPLING TUBE
7.25"
10' COTTON PERMEABLE RETAINER
14' NYLON RETAINER
16' NYLON RETAINER
20' NYLON RETAINER
MINE ROOF
MINE FLOOR

MINE SEAL INSTALLATION

MINE NO.14-042A

HARRISON COUNTY, WEST VIRGINIA

FIGURE 5

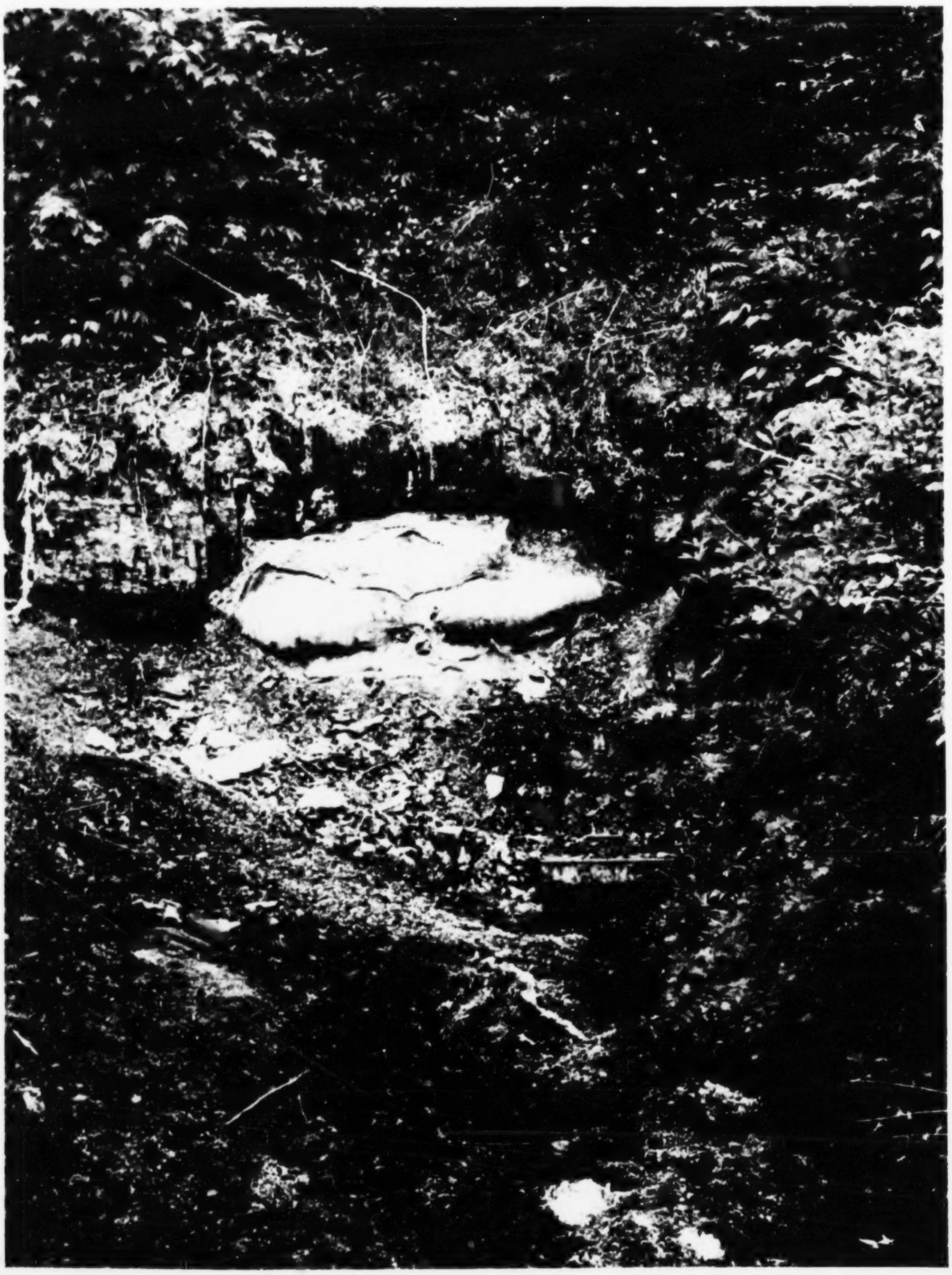


FIGURE 6 - FRONT VIEW-EXPENDABLE GROUT RETAINER SEAL-MINE NO. 14-042A

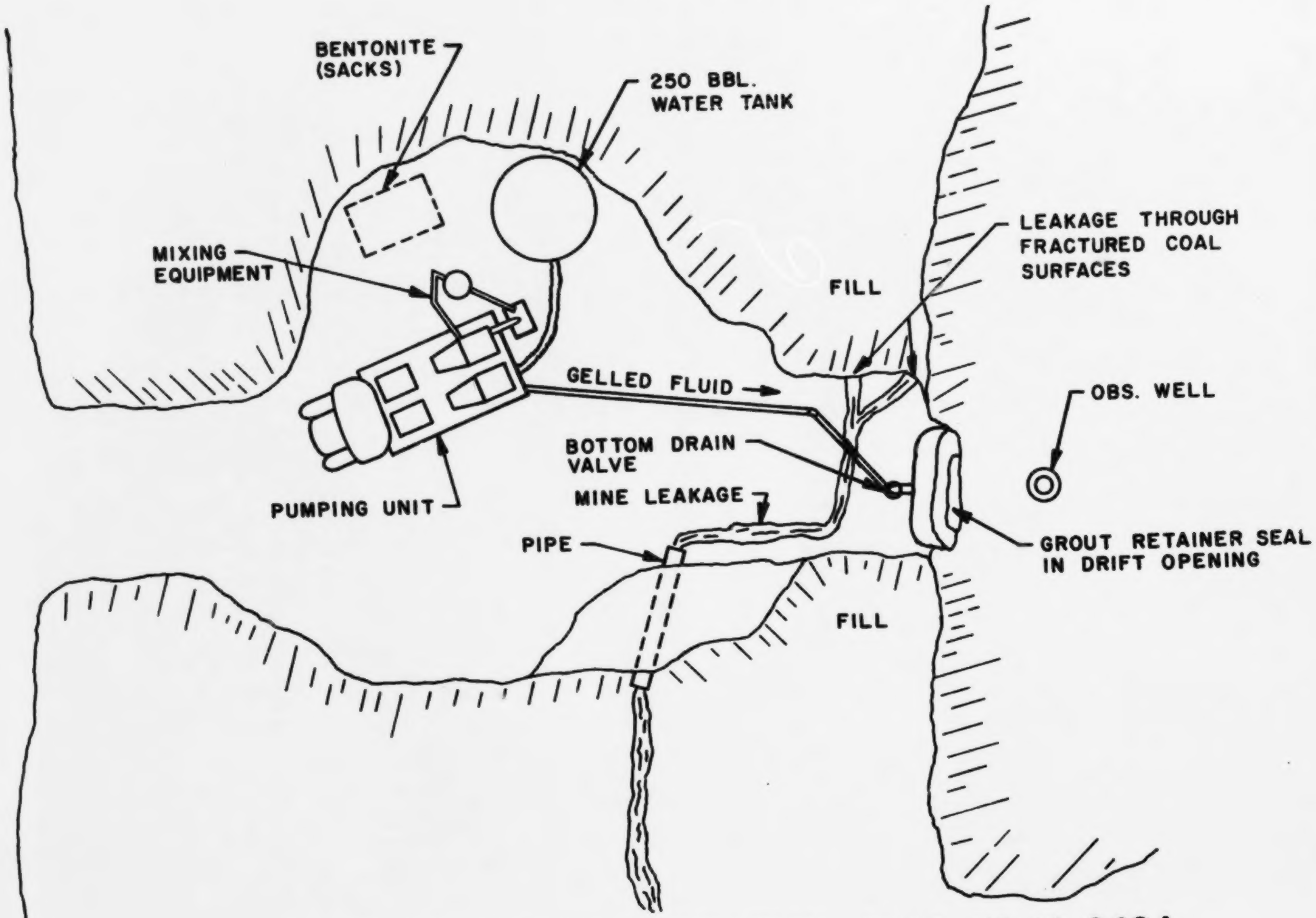

BENTONITE (SACKS)
250 BBL. WATER TANK
MIXING EQUIPMENT
FILL
LEAKAGE THROUGH FRACTURED COAL SURFACES
GELLED FLUID
OBS. WELL
BOTTOM DRAIN VALVE
PUMPING UNIT
MINE LEAKAGE
PIPE
GROUT RETAINER SEAL IN DRIFT OPENING
FILL

FIGURE 7 - FLUID PUMPING OPERATION - MINE NO. 14-042A

showed a depth of 5.0 feet of fluid. The leakage rate was measured at 0.4 gallon per minute.

Equipment utilized for the remedial work is shown in the schematic drawing of the Fluid Pumping Operation attached as Figure 7. This drawing also designates the points of leakage.

The work was initiated by mixing a small batch of gelled fluid consisting of 5 sacks (500 pounds) of Wyoming bentonite and 800 gallons of fresh water. Samples were taken at intervals throughout the job to determine viscosity and filter loss using the standard API testing procedure as given in American Petroleum Institute publication RP 13B, November 1962. A table listing all fluid properties obtained in these tests is included in the Appendix as Table 1.

The fluid was pumped into the mine through the 3-inch drain line which had previously been installed as a part of the mine seal. As the fluid was being pumped into the mine, a fiberous material of shredded cane fiber (Fibertex) was added to act as a sealant for any fractures leaking fluid. The rise in the mine fluid level after the addition of the first batch of gelled fluid was 1-1/4 inches.

After the pilot batch was pumped into the mine, it was decided that four larger treatments would be used for a total of about 40,000 gallons of gelled fluid. Injection of one batch per day was selected. This would allow time for observation and evaluation of each treatment separately.

A batch of 9,500 gallons of gelled fluid was then mixed, using 7,000 pounds of bentonite and 9,200 gallons of fresh water, and pumped into the mine through a 3-inch line. Fibertex was added continuously at the rate of 1/2 pound per 100 pounds of bentonite. This was increased to one pound per 100 pounds after 2,500 gallons had been pumped into the mine. This ratio was then retained for the remainder of the remedial treatment.

Three additional batches were mixed using 10,000 gallons of fresh water and 7,500 pounds of bentonite. These batches, each consisting of 10,300 gallons of gelled fluid and 75 pounds of Fibertex, were pumped into the mine on the next three consecutive days after the first treatment.

During the final pumping stages, some of the viscous fluid was observed seeping through the coal seam on the upper right side of the top retainer. Leakage had not been observed in this area previously. No further leakage was noted after the treatment.

A total of 41,200 gallons of gelled fluid was pumped into the mine, utilizing 30,000 pounds of Wyoming bentonite and 295 pounds of shredded cane fiber blend. A schematic drawing of the mine is attached as Figure 8 to show the condition following the remedial work.

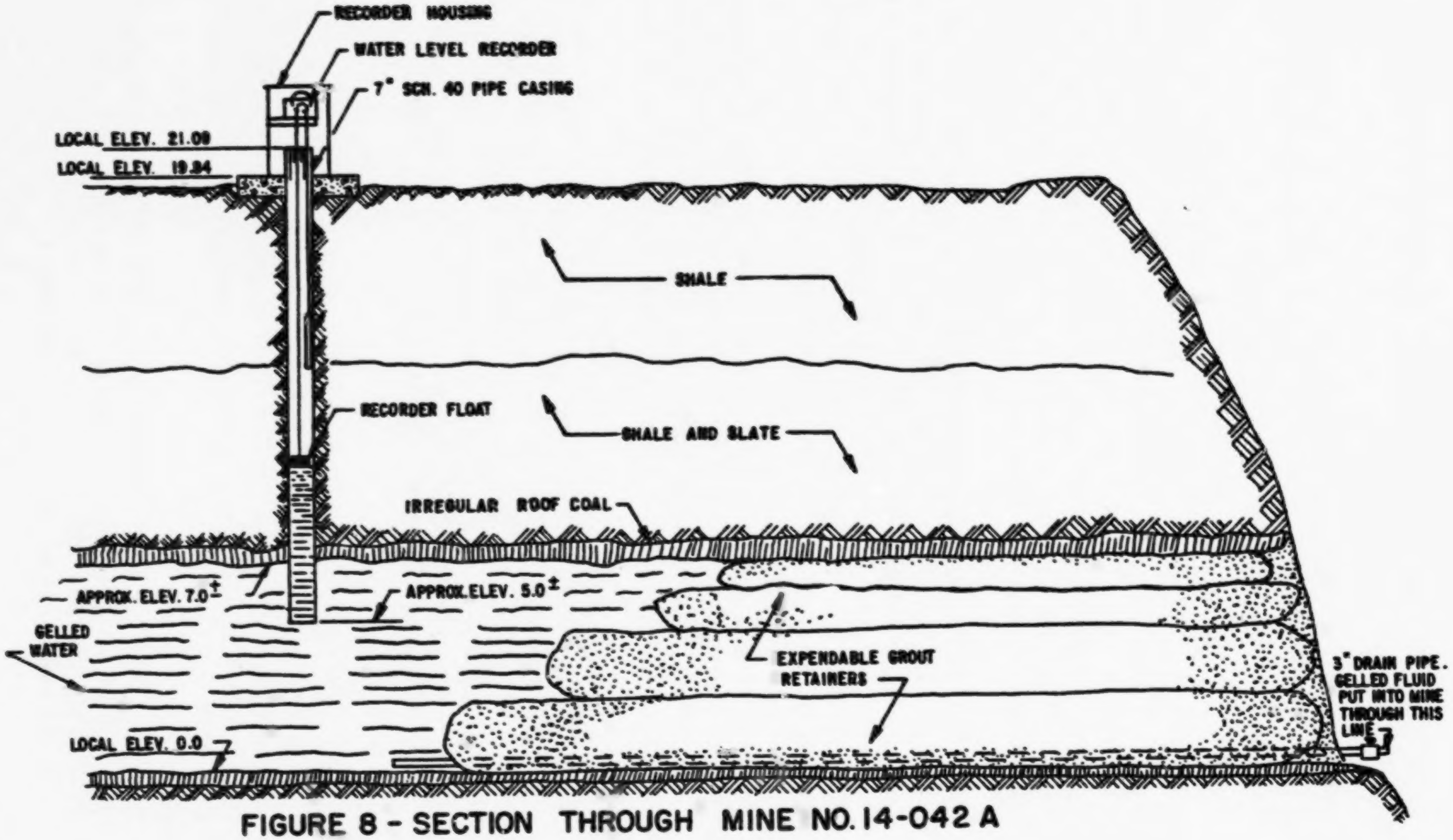

RECORDER HOUSING
WATER LEVEL RECORDER
7" SCH. 40 PIPE CASING
LOCAL ELEV. 21.09
LOCAL ELEV. 19.84
SHALE
RECORDER FLOAT
SHALE AND SLATE
IRREGULAR ROOF COAL
APPROX. ELEV. 7.0±
APPROX. ELEV. 5.0±
GELLED WATER
EXPENDABLE GROUT RETAINERS
LOCAL ELEV. 0.0
3" DRAIN PIPE. GELLED FLUID PUT INTO MINE THROUGH THIS LINE

FIGURE 8 - SECTION THROUGH MINE NO. 14-042 A

The flow rate at the completion of the treatments was 0.55 gallon per minute, an increase of 0.15 gallon per minute over the initial rate. The leakage rate has continued to average about 0.5 gallon per minute since treatment. This is approximately the same as before the treatment, indicating that the treatment did not accomplish the desired results.

The gelled fluid apparently did not fill all the void space behind the mine seal. The rise in fluid level directly behind the seal without any increase in leakage seemed to indicate that the viscous fluid did not travel into the left side of the mine from which the leakage was discharging. A roof fall in that area could have restricted the movement of the more viscous fluid. It is also possible that the water leaking through the coal fractures to the left of the seal might be coming directly from water percolation and not from within the mine. This might also explain the change of leakage rate without any change in water level in the mine. The observation of some gelled fluid discharging from another point during the treatment seems to substantiate the theory that the fluid would discharge through a fracture if the fluid had access to it from within the mine.

The water table within the mine and the discharge rate actually rose about 20% during this time (Table 2 - Appendix). A detailed tabulation of additional monitoring on this mine is included in the Appendix as Table 3 which indicated that this remedial work was not successful in controlling the discharge. Water analyses data are found in Table 4 of the Appendix. It can be seen that the mine water was initially high in acid, iron, sulfate and aluminum content but appeared to be decreasing as time progressed.

The cost for the materials and equipment used in this remedial work on Mine No. 14-042A was $2,771.00. Access rights cost an additional $300.00 and site restoration required an expenditure of $279.00.

Remedial Work - Mine No. 40-016

This is a small, abandoned commercial drift mine located south of Clarksburg, West Virginia, midway between West Milford and Lost Creek in Harrison County. Drift openings can be observed in both the Pittsburgh and Redstone coal seams with the major headways located in the Pittsburgh seam. Both Pittsburgh and Redstone coal have been strip mined.

Mining was begun in this area around 1904 and continued until about 1930. The mine was abandoned but still had some coal reserves present. In addition to the drift and strip mining, extensive augering has been performed in the Redstone coal seam on the south side of the mine island. Any augering which may have been done in the Pittsburgh coal seam is obscured by land reclamation backfill. No subsidence was observed in the overburden above the mine. A geologic section of the mine is included as Figure 65 to show the composition of the overburden.

Drainage from this site discharges into Stone Pot Run and thence to Lost Creek. Flows from 7 to 150 gallons per minute have been measured. The mine water had a pH of 2.9 to 3.4, an acidity from 306 to 396 mg/l in terms of $CaCO_3$ and iron varying from 120 to 190 mg/l.

Two main openings from the Pittsburgh coal were found. These openings are shown on the mine map attached as Figure 9 and designated as Opening No. 1 and No. 2.

In the previous study, Openings No. 1 and 2 were selected for installation of a mine seal. An expendable grout retainer seal was considered for this application, but was rejected when a row of center posts were discovered in Opening No. 2. Based on results of research work, it was decided that limestone aggregate could be placed in the drift opening pneumatically and then consolidated into an impervious mass with a cement grout. This method was used for sealing the two openings of Mine No. 40-016.

Each opening was determined to be 12 feet wide and 7 feet high. Opening No. 1, the main entrance, had a jagged and irregular wall but had no roof falls and was supporting itself well with a minimum of timbering. Opening No. 2, the fanway, had a very rough, irregular floor and roof and a row of center posts. The roof had collapsed and was being supported solely by the timbers for 12 to 15 feet from the opening.

Prior to construction of the seal, the opening was enlarged to full drift size by removing backfill and sloughed highwall. The floor, walls and roof were cleaned by jetting with water with a pressure of 150 psi. This cleaning was performed remotely using a maneuverable jet developed in the previous contract.

To construct the seal, 3/4- to 1-1/2-inch graded limestone was placed in the mine pneumatically. The placement was accomplished remotely using the placement dolly developed in the previous contract. The rock was loaded in pneumatic tanks and moved with air at 25 psi pressure through 4-inch hose to the aggregate placement dolly. The aggregate guide plate on the 4-inch pipe of the placement dolly permitted the operator to remotely control the aggregate placement by rotation of the pipe or movement of the dolly.

The center portion of the aggregate seal contained grout pipes which were installed before placement of the limestone. A layer of minus 3/8-inch aggregate was put on either side of the center section to help contain the grout. The construction details can be noted in the cross section of the installation which is attached as Figure 10. A detailed account of the construction project is given in the Halliburton Company report, Part IV - "Additional Laboratory and Field Tests for Evaluating and Improving Methods for Abating Mine Drainage Pollution", May 21, 1968, p. 126.

After the seals were installed, leakage was noted coming around and through the seals and was also observed across the width of the front base. However, it was reduced to about 7 gallons per minute from both openings, a reduction of 85% from the average of four previous measurements.

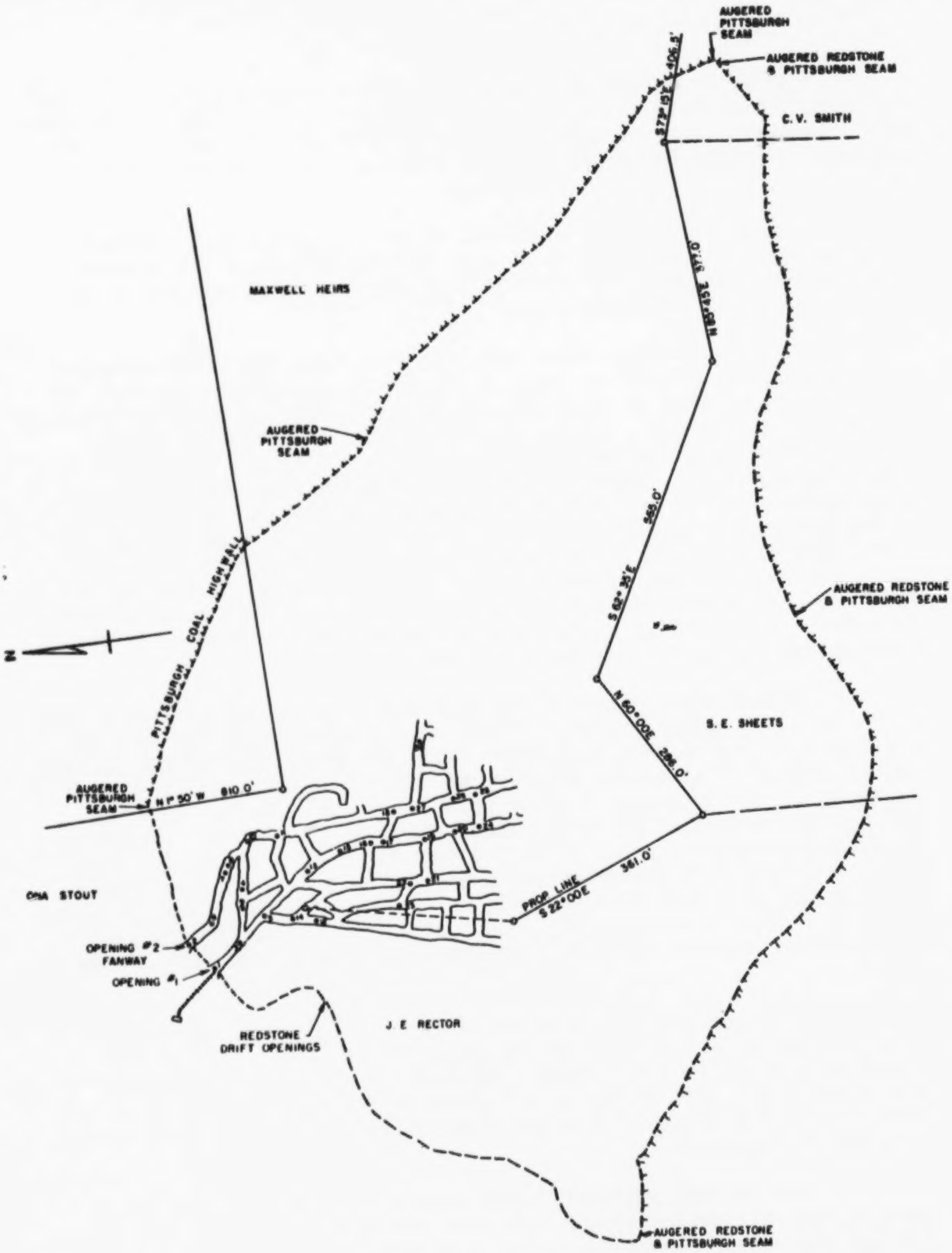
Under the present contract, it was specified that remedial grouting would be performed to attempt to stop or reduce the leakage from the two openings. The main body of the limestone seal was to be grouted using a cement type grout.

The grout pipes which were extending from the existing aggregate seal were uncovered as shown in Figure 11. An effort was made to pump into these pipes to determine if remedial grouting could be accomplished in this manner. It was discovered that the grouting in the previous study had left the pipes plugged so that it was impossible to pump into any of the pipes. It was therefore decided that additional grout holes would be drilled at the face of the highwall on each side of the opening at an angle extending to a point about midway along the length of the aggregate. A crew was retained to survey the site and stake the location for the auger holes into the highwall. Four hole locations were marked and designated for drilling, with two holes into each seal. A plat showing the location of each of these holes as determined by the survey is attached as Figure 12.

Stakes were placed on the base line and a steel pin positioned at the proposed point of entry. These were used to align the hole so that the augering machine could then drill the grout hole at the correct angle. Figure 13 shows the augering machine mounted on a flatbed trailer preparing to drill the No. 1 grout hole. A 6-inch diameter hole was drilled at a height of 84 inches above the base of the coal. Drilling was continued until a depth of 33-1/2 feet had been drilled. At this point, a 2-foot void was encountered and the hole began to seep water. Drilling was continued for an additional 3 feet. At this time, the hole began to produce a considerable amount of water, indicating the possibility that a channel had been encountered leading to the rear of the seal.

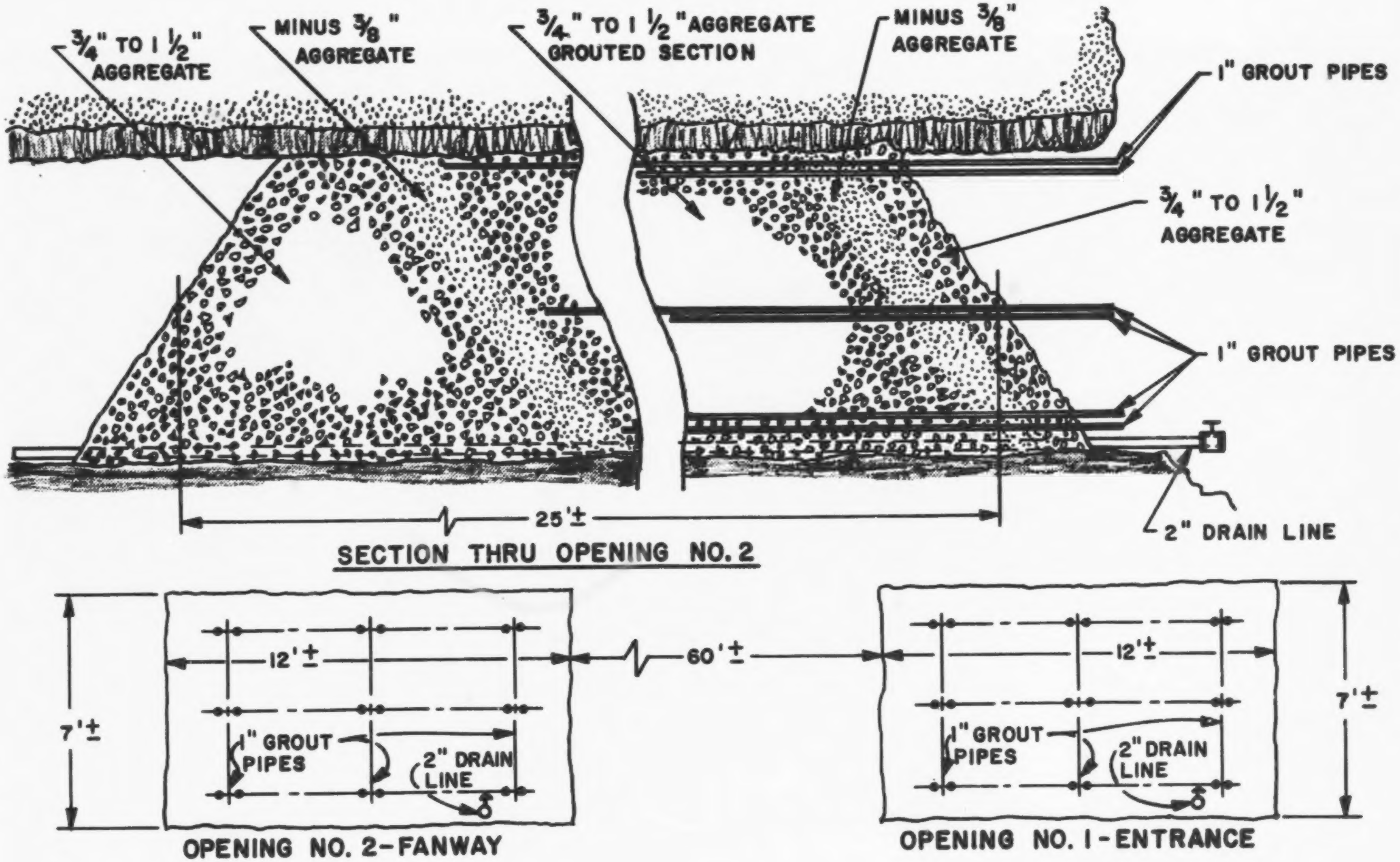
A 2-inch pipe was used in the drilled hole as a grout pipe. A special packer was made to use in grouting the 2-inch pipe in the 6-inch drilled hole. This packer was made by wrapping brattice cloth around the lower part of the 2-inch joint of pipe and banding it on the inward end. The length of the finished packer was 2 feet. This packer is shown in Figure 14.

The 25-foot joint of 2-inch pipe was placed into the grout hole to its full depth, then pulled back approximately 1-1/2 feet to tighten the packer and shut off the water flow around the pipe. A 25-foot joint of 1-inch pipe was then put into the hole alongside the 2-inch pipe until it was at a point about 6 inches from the set packer.


AUGERED PITTSBURGH SEAM
AUGERED REDSTONE & PITTSBURGH SEAM
S 7° 15' E 406.5'
C. V. SMITH
N 80°45'E 577.0'
MAXWELL HEIRS
AUGERED PITTSBURGH SEAM
S 62° 35' E 565.0'
AUGERED REDSTONE & PITTSBURGH SEAM
N
PITTSBURGH COAL HIGHWALL
N 60°00E 298.0'
S. E. SHEETS
AUGERED PITTSBURGH SEAM
N 1° 50' W 810.0'
ONA STOUT
PROP LINE S 22°00E 361.0'
OPENING #2 FANWAY
OPENING #1
REDSTONE DRIFT OPENINGS
J. E. RECTOR
AUGERED REDSTONE & PITTSBURGH SEAM

MINE NO. 40-016

FIGURE 9



SECTION THRU OPENING NO. 2

OPENING NO. 2-FANWAY

OPENING NO. 1-ENTRANCE

3/4" TO 1 1/2" AGGREGATE
MINUS 3/8" AGGREGATE
3/4" TO 1 1/2" AGGREGATE GROUTED SECTION
MINUS 3/8" AGGREGATE
1" GROUT PIPES
3/4" TO 1 1/2" AGGREGATE
1" GROUT PIPES
2" DRAIN LINE
25'±
7'±
12'±
1" GROUT PIPES
2" DRAIN LINE
60'±
12'±
7'±
1" GROUT PIPES
2" DRAIN LINE

FIGURE 10-AGGREGATE PLUG DETAIL-MINE 40-016

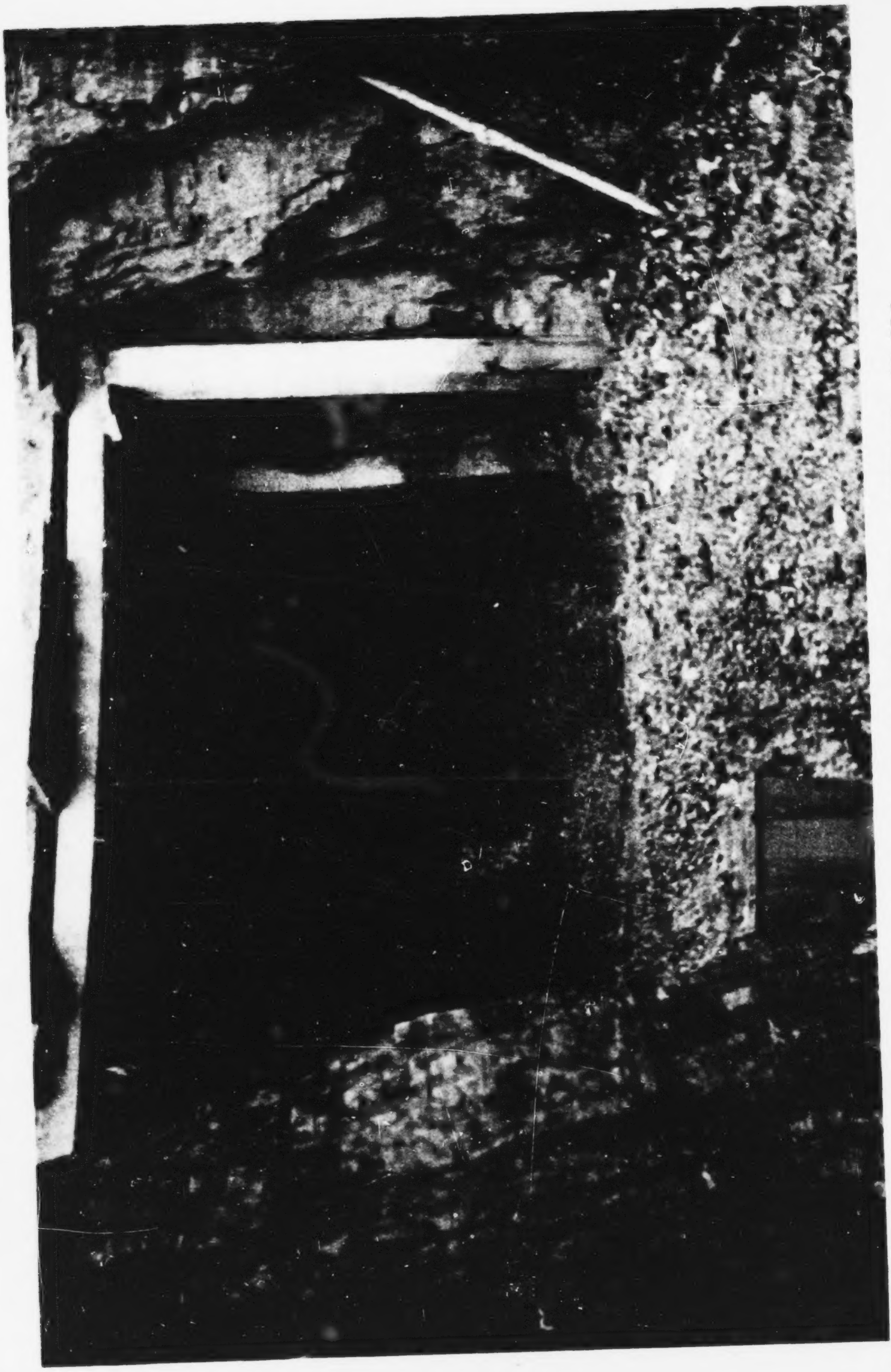


FIGURE 11 - LEAKING GROUTED AGGREGATE SEAL - MINE NO. 40-016

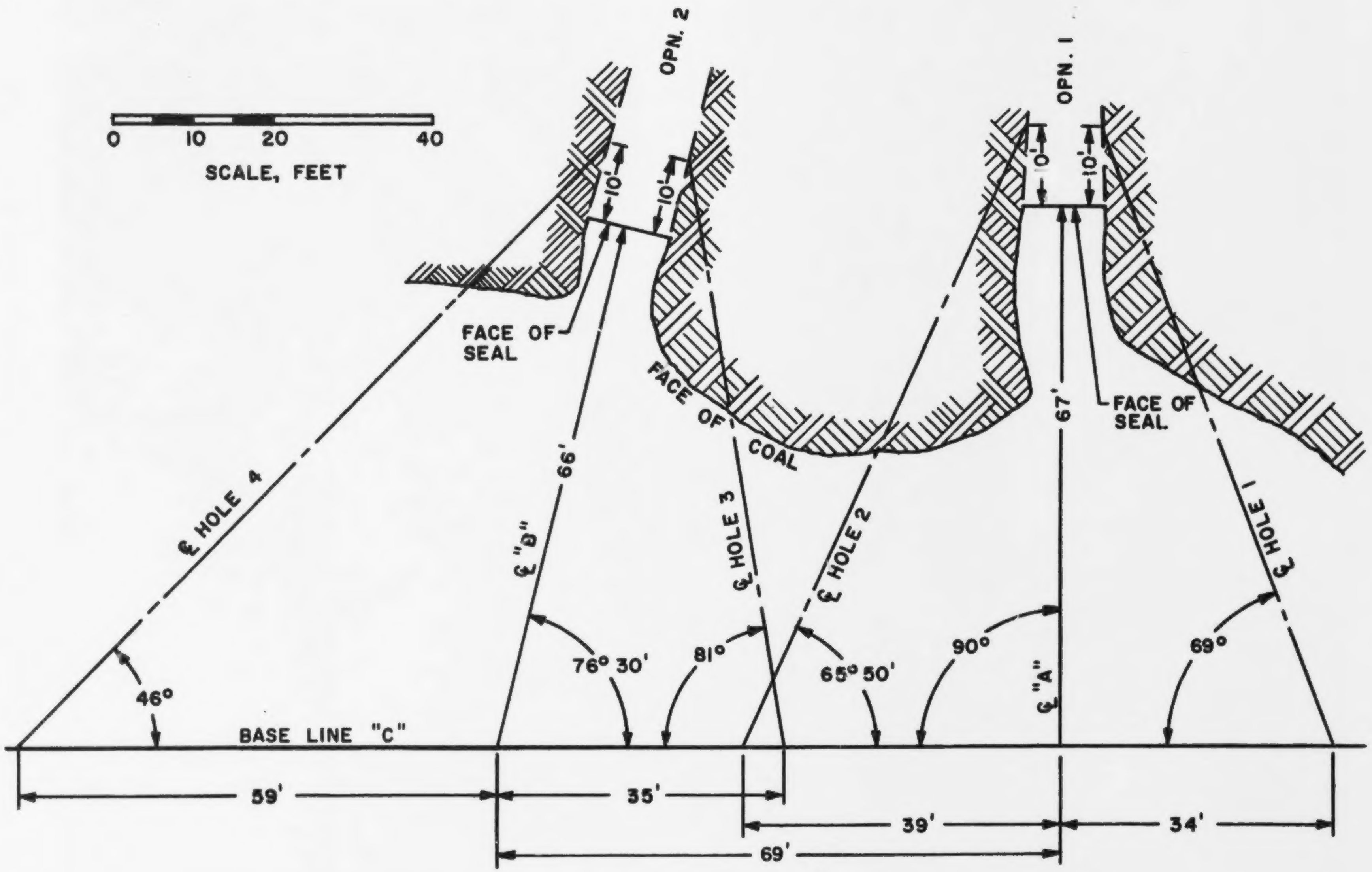

OPN. 2
OPN. 1
0
10
20
40
SCALE, FEET
10'
10'
10'
10'
FACE OF SEAL
FACE OF SEAL
FACE OF COAL
℄ HOLE 4
66'
℄ "B"
℄ HOLE 3
℄ HOLE 2
67'
℄ "A"
℄ HOLE 1
46°
76° 30'
81°
65° 50'
90°
69°
BASE LINE "C"
59'
35'
39'
34'
69'

FIGURE 12 - GROUT HOLE LOCATION - MINE NO. 40-016

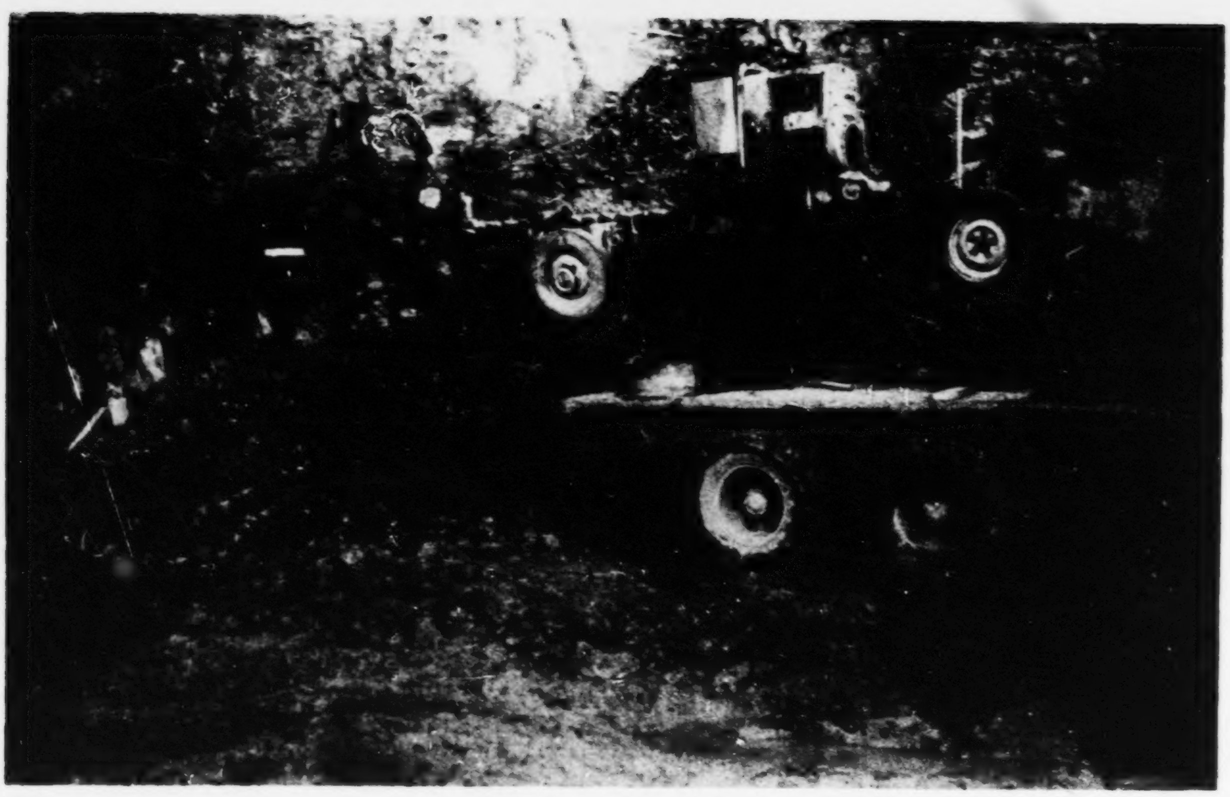


FIGURE 13 - AUGER IN DRILLING POSITION - MINE NO. 40-016

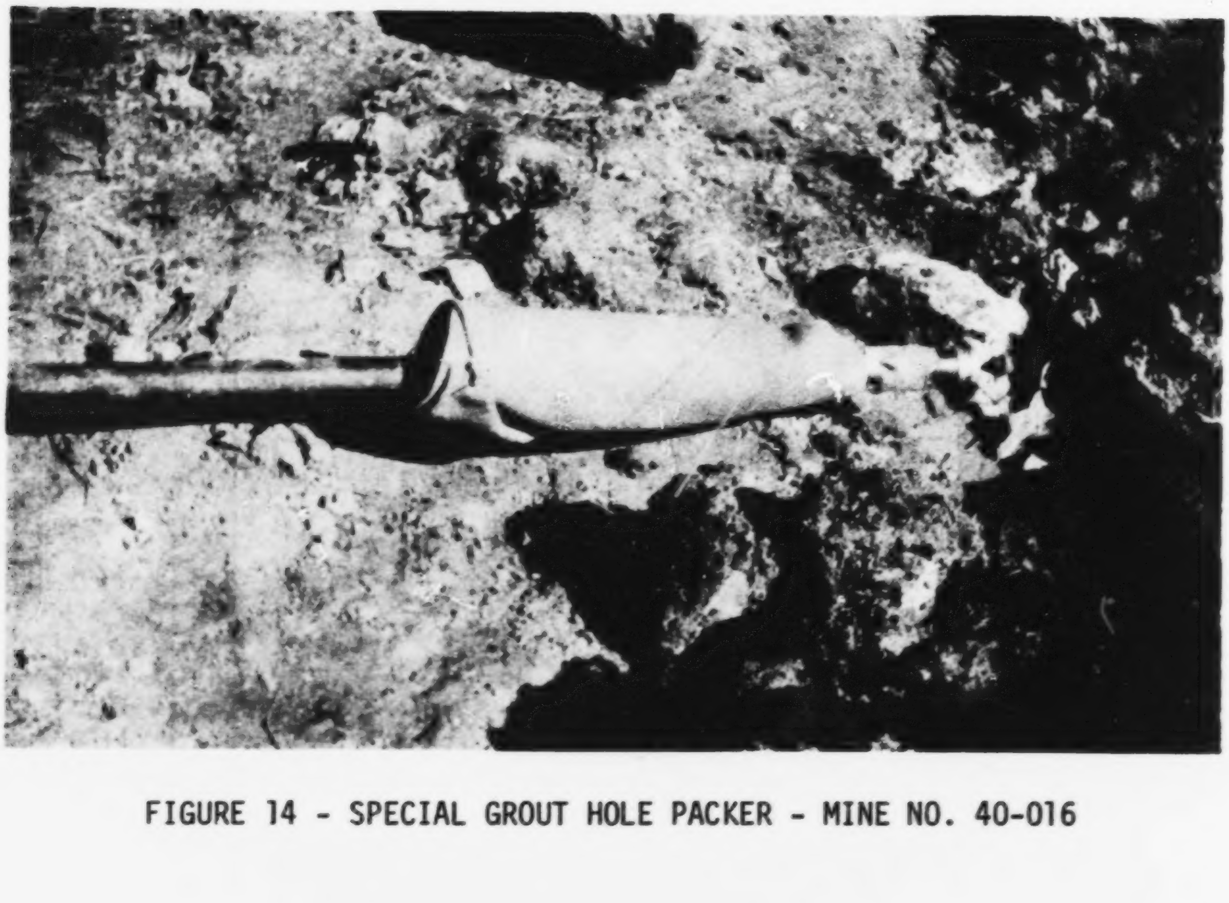


FIGURE 14 - SPECIAL GROUT HOLE PACKER - MINE NO. 40-016

This pipe was used in the grouting of the 2-inch pipe in the hole. A valve was placed on the 2-inch line and closed. This stopped the flow of water from the mine through this pipe.

In like manner, grout hole No. 2 was drilled at a point 56 inches above the base of the coal. The depth of this hole was 45 feet before the hole started discharging water. A 25-foot joint of 2-inch pipe with a cloth packer was placed in the hole in a similar manner to grout hole No. 1. The packer was set and the 1-inch pipe placed alongside the 2-inch pipe for grouting purposes. Grout hole No. 3 was drilled 52 inches above the base of the Pittsburgh coal. Water was encountered at 30 feet, but the hole was drilled to a depth of 42 feet to provide entry into the aggregate plug. The packer was placed on the pipe and set in the hole and the 1-inch pipe was placed alongside for grouting, as was done on the previous holes. The No. 4 grout hole was drilled 57 inches above the base of the Pittsburgh coal. As can be noted on Figure 12, the depth required for this hole is much less than the other holes. Water and limestone chips were encountered at a depth of 23 feet. Drilling was discontinued at a depth of 26 feet. The pipe containing the packer was placed in the hole with the accompanying 1-inch pipe.

The grouting of the 2-inch pipe inside the 6-inch drilled hole was necessary to permit grouting through the pipe. This was accomplished by closing the hole entrance with clay to enclose the 2-inch pipe and a short length of 1-inch pipe at the top of the hole. As cement slurry was pumped into the hole through the long 1-inch pipe, the hole filled completely around the 2-inch pipe until the grout flowed out the short 1-inch top pipe. About 6 sacks of cement were required for each hole to successfully grout the 2-inch pipe.

After allowing sufficient time for the cement grout to harden, the grouting installations were tested using fluorescein dye. In grout hole No. 1, 800 gallons of fluid were injected through the 2-inch pipe at the rate of 20 gallons per minute with 20 psi pressure for the first 400 gallons, and 40 gallons per minute at 100 psi pressure for the last 400 gallons. The return of dye was not evident in the water being monitored from the mine. In grout hole No. 2, 125 gallons of fluid containing dye were injected in 6 minutes. The dye water was observed coming through the coal on the left-hand side of the opening about the middle of the coal vein, extending from the front edge of the aggregate to about 8 feet in front. This seemed to indicate the vertical and horizontal communication that would be prevalent in the proposed remedial work. In grout hole No. 3, 125 gallons of dye water were injected in 8 minutes. The dye water was observed flowing from Opening No. 2 through cavings and debris on the right side and across the top of the aggregate. In grout hole No. 4, 250 gallons of dye water were injected over a 14-minute period. The dye was observed coming across the top of the aggregate seal, covering about three-fourths of the width. Satisfactory returns were obtained from grout holes 2, 3 and 4, indicating that the grout fluid would probably move through the aggregate to the front of the aggregate at the highwall. Grout hole No. 1 did not show any returns, so a grout fluid injected into this hole may not go to the right area to seal off any leakage.

A grout slurry was selected and approved consisting of preblended 50% portland cement and 50% fly ash by volume, with 18% salt and 0.4% Halad-9. Halad-9 is a Halliburton Services additive to prevent water from being lost into permeable formations due to applied pressure. Based on Halliburton's many years of grouting experience, this particular grout slurry was chosen because of its low water loss and good expansive characteristics. This material cost less than one which did not use a blend of fly ash.

A decision was made to begin the grouting operation through grout hole No. 1. A check of the flow through the 2-inch pipe at this hole showed a full stream of water emerging, but preparation continued for using this hole to begin the grouting. The grout slurry was mixed at a weight of 14.5 pounds per gallon using the impounded mine water. The grouting started at a pressure of 10 psi. Approximately 400 gallons of grout slurry were pumped through the pipe when a cement-colored discharge from the 2-inch drain line was observed. The grout slurry was flushed from the pipe with water and the valve was closed.

The water flow through the pipe in grout hole No. 2 was measured at a rate of 2.8 gallons per minute prior to beginning the remedial grouting operation. The grouting was initiated at a pressure of 10 psi. When 140 gallons of slurry had been pumped into the aggregate, the grout broke through the main body of the coal, 10 feet in front of the aggregate seal and 1-1/2 feet from the top of the coal seam. The grouting operation was stopped at that point.

The flow of water through the pipe from grout hole No. 3 was measured at 7 gallons per minute. The grouting again started at a pressure of 10 psi. When 60 gallons of grout slurry had been pumped into the pipe, water discolored with cement emerged through the mud cavings 3 feet up on the coal seam at the front edge of the aggregate seal. Heavy grout started flowing over the top of the aggregate about one foot inside the right-hand row of posts. After pumping a total of 260 gallons of grout, part of which overflowed the top of the aggregate seal, the grouting was stopped.

The pipe in grout hole No. 4 did not have an initial water discharge. Grouting began with a pressure of 15 psi. When 80 gallons had been pumped into the pipe, grout started coming out the top part of the aggregate seal. When 400 gallons of grout slurry had been pumped in, the pumping was stopped.

The grouting operation then moved back to grout hole No. 1. An additional 400 gallons of grout slurry were pumped into the pipe at a pressure of 5 psi. There was a slight discoloration of water coming from the mine drain line, but no other evidence of grout was observed.

Another 400 gallons of grout slurry were pumped into the grout pipe at a pressure of 10 psi. After this operation, the grouting was stopped.

A measurement of the flow from the grout pipes on the day after the grouting operation indicated a flow of 0.35 gallon per minute from No. 1 grout pipe, 0.5 gallon per minute from No. 2 grout pipe and 0.1 gallon per minute from No. 3 grout pipe, and no flow from No. 4 grout pipe. The flow from the mine was 2.2 gallons per minute from the No. 1 Opening and 3.6 gallons per minute from the No. 2 Opening. This was a reduction of about 17% in the total leakage from the mine.

Since a significant reduction in leakage was not obtained, a decision was made to perform another grouting operation using a different type grouting fluid. The grouting material selected was Halliburton's DOC slurry. This slurry consists of cement particles surrounded by a hydrocarbon base fluid incorporating a surface-active agent, or kerosene, cement and a dispersant type surfactant. These materials are mixed to form a slurry that remains inactive unless contacted by water. The slurry will absorb water like a sponge to cause a hard and dense set.

This grouting operation was planned for grout holes No. 3 and 4 on Opening No. 2 because this opening had the larger flow coming from the mine. Arrangements were made to divert the flow from the mine to recover the kerosene and prevent discharge into the streams of the area.

The grouting was begun on grout hole No. 3. An initial flush with kerosene was made at a pressure of 20 psi to remove any water from the pipe to prevent contamination and premature setting of the grout. The special slurry, weighing 15.4 lbs/gal, used in the grouting operation was mixed in 30-gallon batches and pumped through the grout pipe. Ten batches were pumped through the pipe without observation of returns.

Grouting then began on grout hole No. 4 by pumping in 100 gallons of kerosene at 50 psi pressure as a spearhead, followed by a 30-gallon batch of the special grout. This was pumped into the pipe at a rate of 20 gallons per minute. Cement returns were noticed immediately coming over the top left of the aggregate seal, so the line was flushed with kerosene. Another batch of the slurry was mixed at the same weight and injected. The leak at the top of the aggregate was closed off, but leakage was still observed on the lower right. Later, two other batches were injected at a pressure of 150 psi without any cement returns being observed. A batch was then pumped in at a weight of 15.6 lbs/gal and still no cement returns were observed. Another batch of 30 gallons was pumped in with the pressure increased to 250 psi. No returns were seen. An attempt was made to pump in another batch of 30 gallons, but the pipe would not accept the grout slurry.

The connection was moved to grout hole No. 3 and the remaining grout pumped in at this point, followed by another 30-gallon batch. No visible returns were noted. Therefore, the grouting operation was halted at this point.

The flow rate on the two openings was measured on the following day at 2.0 gpm for Opening No. 1 and 2.2 gpm for Opening No. 2. This is a reduction of 27% from the flow prior to the second grouting operation, or a 37% reduction from the original leakage.

Although the leakage had been reduced, it was decided to try an additional grouting operation using the regular cement-fly ash mixture which had been used in the first operation, with the addition of Flocele, a shredded cellophane, as a plugging material. A total of 2,400 gallons of this material containing 75 pounds of Flocele was pumped into Opening No. 1 through grout holes No. 1 and 2 at a maximum pressure of 50 psi. A slight discoloration was noticed from the drain line, but no cement grout returns were observed. The leakage from Opening No. 1 after the grouting was measured at 1.6 gpm. On the following day, the flow rate from Opening No. 1 was 2.12 gpm and from Opening No. 2 was 2.41 gpm. There was essentially no reduction of leakage due to this grouting.

Monitoring of the flow from the openings of Mine No. 40-016 was continued at regular intervals throughout the duration of the project. The water data are shown in the Appendix (Table 5). It can be noted that the level of the fluid impounded behind the seal is higher in Opening No. 1 than in Opening No. 2, but the discharge rate is higher from Opening No. 2. This could be a result of the bad condition of Opening No. 2 prior to the original seal installation, particularly the floor and roof areas, which would make a seal much more difficult to obtain. The fluid level of impounded water increased over 10% during the 11 months of monitoring after the remedial grouting. Two months after grouting, the total leakage returned to a point approximately equal to that measured prior to grouting, possibly due to the increased head. However, before the grouting was performed, the average acidity value for Opening No. 1 was 267 mg/l, while the average after grouting was 150 mg/l. A similar reduction from 200 mg/l to 96 mg/l was noted for Opening No. 2. This indicates that some channels of leakage around the plug were sealed by the grouting and the leakage was channeled through the limestone where some neutralization occurred.

The analyses of the samples from the two openings are listed in the Appendix (Tables 6 and 7). All samples show a relatively high acid and iron content, even when the pH varies from 3.1 to 6.7 from the same opening. It should be noted that the pH increased after grouting. The average pH of the discharge from Opening No. 1 increased from 4.1 to 5.3 and from 5.1 to 6.3 at Opening No. 2. This is in agreement with the decrease in acidity mentioned above.

After grouting the seal and causing the water to flow through the limestone, the acid production rate from Opening No. 1 decreased from 64 pounds per week to 37 pounds per week. This is a reduction of 42%. In a similar manner, the acid production rate from Opening No. 2 was reduced 64%, or from 83 pounds per week to 30 pounds per week. The total acid production from both openings was reduced 54% by the grouting.

The cost for materials and equipment used in the remedial work on Mine No. 40-016 was $6,007.00. The drilling and preparation of the grout holes accounted for $1,468.00 of this cost. Site preparation and restoration required an additional expenditure of $1,310.00. The cost for the grout material pumped in place was determined. Exclusive of costs for drilling and preparing grout holes and the site preparation and restoration, the cement grout cost $0.56 per gallon in place and the same material containing Flocele was $0.57 per gallon. The DOC grout cost was $3.41 per gallon in place.

BULKHEAD DEVELOPMENT

Improvement of Aggregate Placement

During the previous study, research was conducted by the Halliburton laboratories at Duncan, Oklahoma on movement of various sizes of aggregate using air as the transporting medium. The aggregate sizes used in the tests were 3/4-inch and 3/8-inch graded limestone. The aggregate was moved from one pressure tank to another tank through 150 feet of 5-inch inside diameter pipe and hose. The pressure vessels were 200-cubic-foot tanks with a rated working pressure of 40 psi. A compressor rated at 465 scfm at 40 psi was used for the air supply.

The 3/4-inch limestone was conveyed from tank to tank at a rate of approximately 600 pounds per minute at a pressure of 20 psi. The 3/8-inch aggregate rate was 627 pounds per minute. This rate was only for the actual conveying and did not include any time for reloading. The rock was also conveyed a distance of 23 feet vertically during these tests.

Additional tests were conducted at Weston, West Virginia using truck mounted pressure tanks. These tests did not establish a rate, but did confirm the feasibility of using this equipment for a field test.

A field test was performed using the two openings of Mine No. 40-016. For the placement of the aggregate in these openings, a 150-cubic-foot truck mounted pressure tank was utilized. The tank was loaded by a belt conveyor. The conveyor was supplied by a front-end loader from a pile of aggregate on the work site. Loading of the tank was accomplished at a rate of 1,000 pounds per minute for the 3/4- to 1-1/2-inch aggregate. An overall average of 500 pounds per minute was pneumatically conveyed into the mine drift. An initial air pressure of 35 psi was used in the pressure tank. This pressure reduced to 10 psi as the aggregate moved through the line.

In addition to the time required for loading which stopped the conveying, a problem was encountered when trying to move wet aggregate. The limestone movement through the pipe was decreased in proportion to the fineness of the aggregate and the moisture present. It was found that the minus 3/8-inch aggregate would tend to bridge because the moisture would cause the fines and small rocks to bind together and prevent movement, thereby forming a plug in the lines. It was also determined that the rate and distance any aggregate could be moved was in inverse proportion to its size.

The purpose of this phase of the project was to develop a system for placement of the aggregate into a mine in a more efficient manner than accomplished in the prior field tests. This was to be done by developing a continuous handling system or a system with a minimum downtime required for reloading.

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The first tests conducted were with 1-1/2-inch (No. 3) aggregate. The pressurized tank was used without any modification. The aggregate was loaded in one 160-cubic-foot pressure tank and attempts were made to blow the material to an adjacent tank through 20 feet of 4-inch line. Air pressure of 30 psi was supplied by one compressor of 175 cfm capacity. Results were poor, so an additional compressor of like capacity was added. The rock still did not move well. Additional air was added up to 675 cfm. Rock moved slowly and bridged frequently in the line, so testing of this size rock was discontinued.

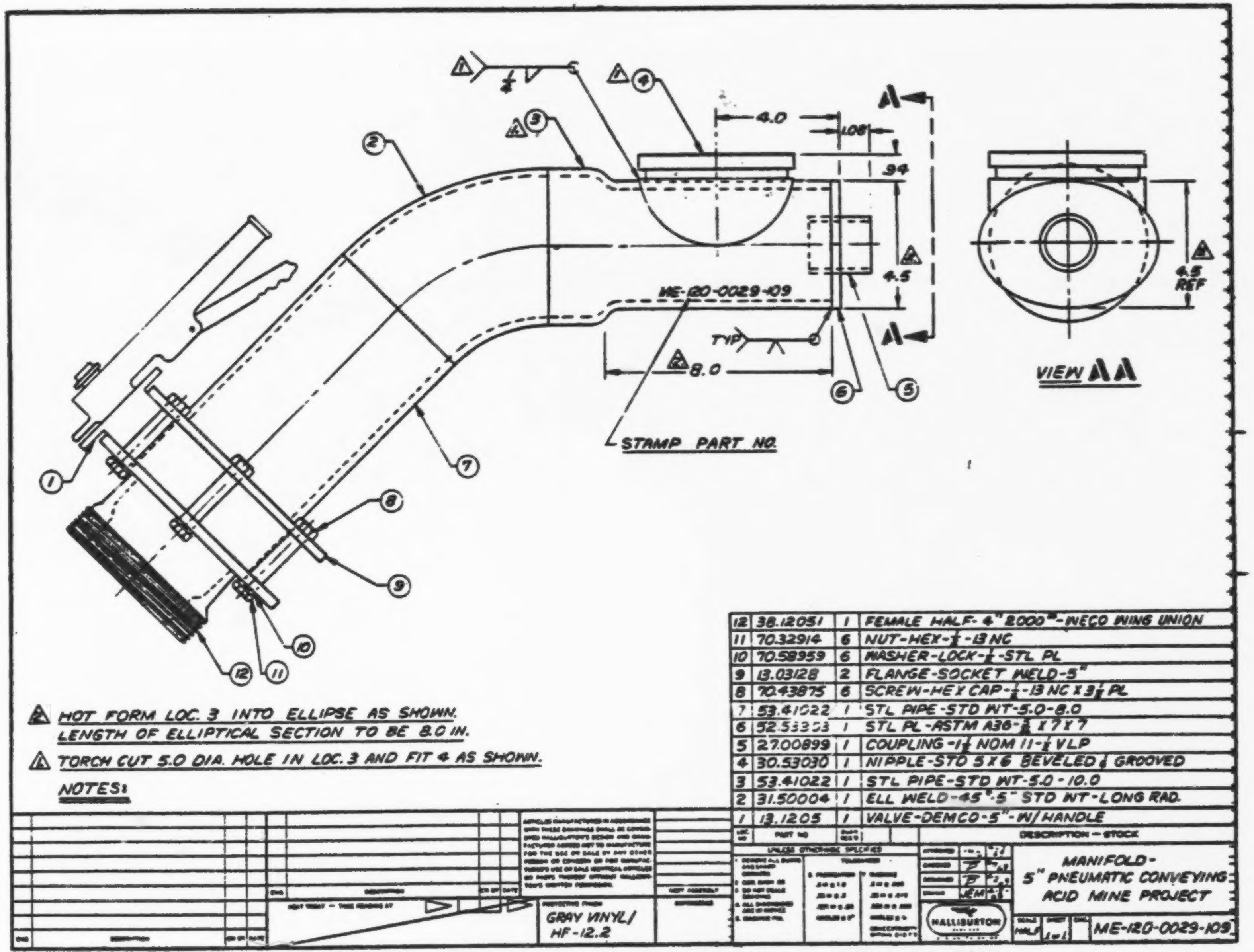
A special bottom discharge manifold was then built to fit the bottom of the pneumatic tank used in these tests. This manifold is shown in Figure 15. It incorporated a connection for an air jet which permitted air to be injected at the point where bridging was most likely to occur.

New tests were arranged, using a 160-cubic-foot pressure tank which had been modified by attaching the special bottom discharge manifold. A view of the test site is shown in Figure 16. The aggregate was moved back and forth between the two tank units. The amount moved was determined by weight of the unit on the left, which was located on a scale.

Thirty tests were made using conveying lines of 40, 60 and 80 feet, with initial tank pressures of 15 and 20 psi. Conveying rates varied from 128 to 1,081 lbs/min with an average rate of 604 lbs/min. This inconsistency in the rate, particularily at the conveying length of 80 feet, was due to several reasons. The tests were made over a period of seven days under various weather conditions, including freezing temperatures which caused the aggregate containing moisture to freeze and bridge in the tank discharge. The first 23 tests were made using the same batch of aggregate. Each test caused some wear on the rock and produced more fines. The finer aggregate then moved at a faster rate. Some of the rate variation was due to changes in compressor speed, which altered the air output accordingly. Higher rates were obtained from one unit then when conveying from the tank on the other unit, showing that the tank condition affects the discharge rate. Detail test data are shown in Table 8 in the Appendix.

An investigation was made to determine if the pneumatic conveying of the aggregate could be a continuous process over a long period of time. A method using a hopper over an air line was considered, but the idea was discarded when it was determined that previous tests had already proven this method to be ineffective. A vane-type feeder was also considered, but was rejected because of air leakage around vanes and large aggregate size which created further sealing problems at the vanes.

Since a method of continuous operation did not seem to be feasible, plans were made for a test which would have the least possible downtime. The test equipment included two mobile units having 160-cubic-


1
2
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11
12
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4.0
.108
.94
4.5
4.5 REF
8.0
TYP
ME-120-0029-109
STAMP PART NO.
A
VIEW AA
HOT FORM LOC. 3 INTO ELLIPSE AS SHOWN. LENGTH OF ELLIPTICAL SECTION TO BE 8.0 IN.
TORCH CUT 5.0 DIA. HOLE IN LOC. 3 AND FIT 4 AS SHOWN.
NOTES:
GRAY VINYL/ HF-12.2
MANIFOLD - 5" PNEUMATIC CONVEYING ACID MINE PROJECT
HALLIBURTON
HALF
1 of 1
JEM
ME-120-0029-109

LOC. NO.	PART NO.	QUAN. REQD.	DESCRIPTION – STOCK
12	38.12051	1	FEMALE HALF-4" 2000#-WECO WING UNION
11	70.32914	6	NUT-HEX-½-13NC
10	70.58959	6	WASHER-LOCK-½-STL PL
9	13.03128	2	FLANGE-SOCKET WELD-5"
8	70.43875	6	SCREW-HEX CAP-½-13 NC X 3½ PL
7	53.41022	1	STL PIPE-STD WT-5.0-8.0
6	52.53303	1	STL PL-ASTM A36-½ X 7 X 7
5	27.00899	1	COUPLING-1½ NOM 11-½ VLP
4	30.53030	1	NIPPLE-STD 5X6 BEVELED & GROOVED
3	53.41022	1	STL PIPE-STD WT-5.0-10.0
2	31.50004	1	ELL WELD-45°-5" STD WT-LONG RAD.
1	13.1205	1	VALVE-DEMCO-5"-W/HANDLE

FIGURE 15 - BOTTOM DISCHARGE MANIFOLD - PRESSURE TANK

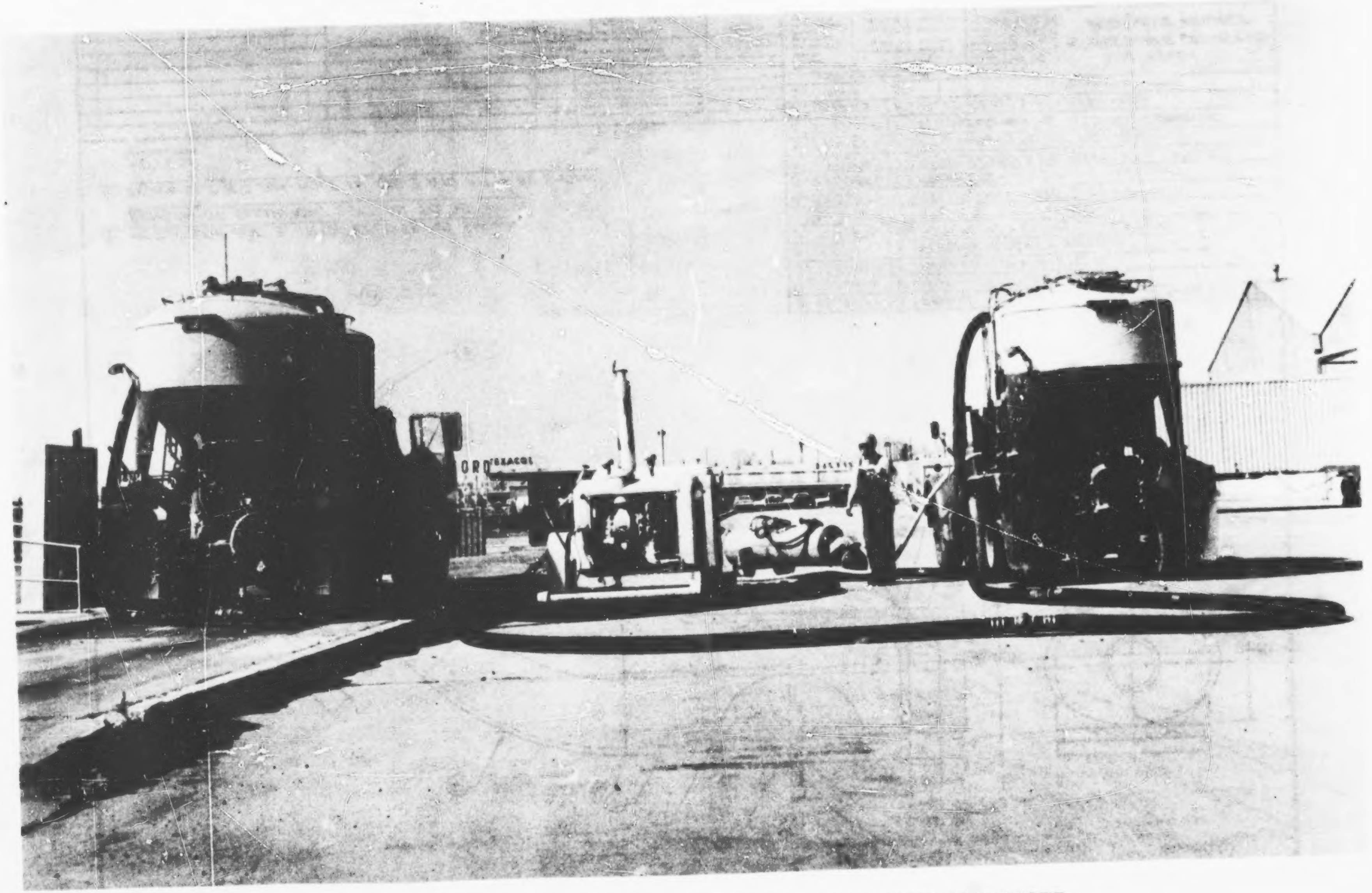


FIGURE 16 - PNEUMATIC CONVEYING TEST - 3/4-INCH AGGREGATE

foot pressure tanks. The loaded tank was then moved forward so the second unit could move under the conveyor to be loaded while the first tank contents were being blown into the receiving trailer. The process was then repeated, reloading the first tank while the second tank discharged.

The test was initiated with the first tank containing 14,570 pounds of 3/4-inch aggregate and the second tank containing 11,200 pounds. Material was conveyed from the first tank through 80 feet of 4-inch hose to the receiving trailer in 25 minutes. Initial tank pressure was 25 psi and 11 to 12 psi was maintained during time of discharging with one 125-cfm compressor. The other two compressors were discharging into the bottom discharge manifold.

Six minutes were required to make the necessary connections to the second tank and initiate movement of aggregate. The first tank was reloaded while unloading the second tank. Eight minutes were required to reload the first tank with the belt conveyor. Twenty-four minutes were consumed in discharging aggregate from the second tank.

The reloaded tank was then discharged, requiring six minutes for connections to be made and 15.65 minutes to move 11,150 pounds of aggregate through the 80 feet of 4-inch hose.

In this test, 36,920 pounds of aggregate were moved 80 feet pneumatically in 64.65 minutes with 12 minutes downtime between runs. The total time of 76.65 minutes gives an equivalent rate of 481.7 pounds per minute for the entire test.

One additional test was made using this procedure while building a bulkhead in a simulated mine fixture. This test is described in another part of this report. During this test, six tank loadings were made using a total of 66,220 pounds of 3/8-inch (No. 12) aggregate. Conveying time for the six tanks of aggregate was 79.6 minutes. Time for loading was only 24.3 minutes, or about 4 minutes average per tank. The time for making connections was 8.7 minutes per load. This test moved the aggregate at a rate of 544 pounds per minute. This is a faster rate than was obtained in tests with the larger limestone.

Tests were conducted at tank pressures of 15 psi and 20 psi. Results confirmed prior experience of Halliburton. As pressure increased, the conveying rate of the aggregate increased. However, for each size conveying line a maximum pressure was obtained for the maximum flow through that size line without stoppages. Increasing the pressure results in bridging of the flow. The use of a larger conveying line would permit higher tank pressures and a subsequent higher discharge rate.

Moisture in the aggregate would reduce the discharge rate and tend to clog the conveying line. Attempts to measure the moisture content accurately were relatively unsuccessful since a representative sample was difficult to obtain and transport to a laboratory without losing moisture. A field sample sent to the Duncan laboratory showed a moisture content of 3.57%, but the accuracy is questionable due to the time and distance involved.

Tests conducted show that the conveying rate varies inversely in proportion to aggregate particle size. Limestone was tested from 1-1/2-inch to 3/8-inch in size to obtain this information. The 3/4-inch size aggregate was moved 80 feet during these tests and up to 150 feet in previous tests. Smaller aggregate can be moved even further.

Since there are several variables in pneumatic conveying, the movement of aggregate can be accomplished almost as desired by adjusting the variables to meet the prescribed conditions.

Quick-Setting Mine Bulkheads

The purpose of this phase of the project was to develop some type of bulkhead which could be placed remotely, used under variable mine conditions, and offer a significant reduction in mine sealing costs.

The use of an expanding foam material was proposed as temporary front and rear bulkheads to contain a low-cost filling material. A feasibility study was then conducted to investigate organic foam materials such as polyurethane and urea-formaldehyde.

A polyurethane foam (Foam A) was considered, based on available published literature and company experience on other projects. Chemical and physical properties appeared to be satisfactory (Table 9 - Appendix). However, uncontrollable mine atmosphere and temperature could adversely affect the density of this foam. Simulated test equipment required was studied and found to be expensive. Material costs were estimated to be $1,000.00 for each of the two necessary bulkheads to contain a filler material in constructing the seals, assuming the foam was applied at 3 pounds per cubic foot density. Polyurethane foam was not considered further as a bulkhead material in mine sealing applications.

Urea-formaldehyde (Foam B) properties were considered and are also listed in Table 9. Such foam material was developed primarily for use as thermal insulation in building construction. It is less dense and has lower compressive strength than polyurethane foam. Also, urea-formaldehyde resin foams are susceptible to biological and dilute acid attack, as well as being destroyed by contact with water. Foam generating units, available from several manufacturers, range in price from $2,500.00 to $3,000.00. Although biological, acid, and/or water attack would probably not be swift enough to prevent urea-formaldehyde foam from serving as a temporary bulkhead, these factors together with low strength, density and high equipment costs were responsible for abandonment of further consideration of the foam in this application.

Inorganic foam materials were investigated in the laboratory and several formulations were tested (Table 10 - Appendix). Formaldehyde, hydrogen peroxide and sodium silicate were used in each case together with a filler material such as silica flour or fly ash. The first foams prepared were very fragile and crumbled easily under finger pressure. Although foam strength increased as the investigation proceeded, water permeability was found to be high. Also, hydrogen gas was liberated in all formulations and could create potentially explosive mine atmospheric conditions. Formaldehyde could cause uncomfortable working conditions for personnel. No further consideration was given to these inorganic foaming materials.

Research effort was then directed toward the development of a material which would be quick-setting and self-supporting with sufficient strength to withstand expected forces. Initial work was centered around the use of various types of accelerators with cementitious materials.

Previous experience with cementing materials dictated the use of bentonite in the mixtures to obtain better thixotropic and viscosity characteristics. By prehydrating the bentonite and mixing it with a slurry of cement, flocculation would occur and give higher viscosity. Gypsum was known to achieve a fast set and impart viscosity, so it was decided to incorporate gypsum into the tests.

Various slurries were prepared in the laboratory using combinations of portland cement, fly ash, gypsum, bentonite, calcium chloride and salt. Ratios of water and other materials were adjusted in an attempt to obtain the most desirable qualities. Twenty-four different slurries were tested in the initial laboratory tests. Slurries were checked for mixing characteristics, angle of repose and set time. All of the slurry compositions tested and reasons for either rejection or selection for further consideration are listed (Table 11 - Appendix).

Eight slurries were selected with characteristics which would be desirable for use in construction of a bulkhead or seal in a mine drift. These slurries were given additional laboratory mixing tests and compressive strength tests. Compressive strength specifications were set for a strength of 10 psi at 70°F within one hour after mixing, and a strength of 300 psi at 70°F after 48 hours. This data is shown in Table 12 in the Appendix.

Because of the quick set of the slurries, it was necessary to depend on visual examination to determine how viscous each slurry was as it was mixed. Slurry No. 13 appeared to be the most viscous, but was expensive and would have contaminated water with diesel oil used in the slurry and did not have the required compressive strength. Slurry No. 9 was viscous but did not have the required compressive strength. This would have made the control of the pumping and mixing very difficult. Slurries No. 12, 21 and 22 were also considered sufficiently viscous.

Cost for materials, exclusive of any charges for handling, blending, mixing or pumping, ranged from $14.00 per cubic yard for Slurry No. 9 to $32.43 for Slurry No. 6. From these slurries, Slurry No. 22 was selected for larger scale testing. This slurry was composed of 60 parts cement, 40 parts gypsum, 6 parts bentonite and 100 parts water by weight, and had a cost of $18.00 per cubic yard for materials.

In a simultaneous research effort, the use of portland cement with various combinations of sodium silicate, silica flour, bentonite, sand and fly ash was being studied (see Table 13 - Appendix). The primary problem encountered with these slurries was the high degree of fluidity which caused a low angle of repose. It can be noted that formula 1414-13-1 costs about one-third the cost of formula 1414-18-1. However, the most expensive formula develops sufficient strength to support the weight of a man in less than one minute, while it requires several hours for the other slurry to develop a similar strength.

Tests were initiated with full-size pumping and mixing equipment using slurries developed in laboratory tests. The slurries were designed as two-stream systems, so a special mixing head previously designed by Halliburton engineers was used to blend the two slurries. This head was a 5.5-inch diameter pipe, 18 inches long, with a 9-inch diameter jacket around it. Slots in the inner pipe enabled fluid entering from each side of the jacket to mix as they entered. The manifolding arrangement on the tests was such that the two fluids could be mixed together before they entered this jacket to provide longer mixing time. This was the procedure used in the initial tests. Both the gypsum cement slurry and the sodium silicate cement slurry were tested in this system.

The first tests indicated that sufficient mixing was not being obtained through the mixing head. The mixing system was changed from 2-inch piping to 1-1/2-inch piping. The mixing head was removed. A perforated disc was placed in the line so the slurries would pass through it immediately after the two streams blended. Various lengths of pipe from 5 feet to 63 feet were then added to the discharge system during the tests. A better buildup of material was obtained with the longer pipe which afforded additional mixing.

Twenty-eight full-scale test series were conducted in developing the correct slurry, placement equipment and techniques. Various ways of mixing the components were tested for both the gypsum cement and silicate cement slurries.

After twelve test series, an evaluation of the two general types of slurries revealed the following information:

(1) The gypsum cement type slurry had excellent viscosity properties and built up readily into a wall. However, the slurry did not set fast enough. When the wall reached a height of about 12 inches, the weight caused the lower part to

slide. Figure 17 shows the good viscosity characteristics of this slurry.

(2) The silicate cement slurry set very quickly, but would not develop a viscosity sufficient to allow a wall to be built.

In view of the above findings, it was decided to try to combine the good viscosity qualities of the gypsum cement and the fast set of the silicate cement. This resulted in many more tests, varying the amount of sodium silicate, the percentage of bentonite and the volumetric ratio of the cement slurry to the bentonite-sodium silicate slurry. It was also determined that the amount of mixing and the length of time between mixing and discharge had a decided effect on the end product. It was found that viscosity could be controlled either by changing the length of pipe after the mixing or by varying the pumping rate. This proved to be an excellent control factor when building a bulkhead.

The full-scale tests also were an aid in developing the proper mixing and discharge equipment. After eight test series, the discharge was changed from open-end tubing to a flattened nozzle about 1 inch by 4 inches, as shown in Figure 17. This nozzle plugged when using high viscosity slurries. Two other variations of the flattened nozzle were tested with similar results. Three additional nozzles were designed and tested to provide a good discharge pattern and eliminate plugging problems. Two of these were fabricated from tubing weld elbows. The third nozzle was machined. A test of the machined nozzle, shown in Figure 18, shows the excellent discharge pattern and coverage obtained. This design was selected for use in the remainder of the laboratory and field tests because it could be used in a 360° arc, it sprayed material in a straight path as it was rotated and it showed no tendency to plug with heavy slurries. A drawing of this nozzle is included as Figure 19.

After five additional test series had been made, a suitable slurry had been developed composed of cement, bentonite, sodium silicate and water that had an instantaneous viscosity sufficient to build up properly and a set time of approximately 12 seconds. The bentonite and sodium silicate slurry were mixed in a two-stream system with cement slurry to produce the finished slurry. The cement slurry was cement and water with a water-cement ratio of 0.43 or 4.85 gallons per sack. The other slurry was composed of bentonite, sodium silicate and water. The bentonite was 10% by weight of water used. It was prehydrated into the water and then the sodium silicate was added. Tests proved that a satisfactory ratio of bentonite slurry to sodium silicate was 3 to 1, or the volume of sodium silicate would be one-third that of the bentonite slurry. When the two slurry streams were mixed, the resultant slurry characteristics varied according to the proportion of the mixture. A greater amount of cement slurry caused higher viscosity while an increase in the silicate gel slurry caused a faster set. After evaluation of the test results, it was determined that a 1 to 1 volumetric ratio of the two slurries would produce the best effect.

A compressive strength test of the selected slurry indicated a strength of 100 psi in 2 hours and 555 psi in 48 hours. The cost of this slurry for materials only was calculated to be $20.90 per cubic yard. Detailed data for this slurry are shown in Table 14 in the Appendix.

A simulated mine drift was built for a full-scale test. The drift, constructed of wooden framing and plywood, was 10 feet wide, 7 feet high and 8 feet long. Three wooden posts were set inside the drift on either side. A portion of the simulated mine drift is shown in Figure 20. This picture also shows the beginning of the bulkhead which was built in the opening.

The slurry used for this bulkhead was the selected sodium-silicate type cement developed in previous tests. A batch of 15 cubic yards was mixed, using the following components:

Slurry 1	
Water	870 gallons
Cement	180 sacks

Slurry 2	
Water	1,125 gallons
Bentonite	945 pounds
Sodium Silicate	375 gallons

The two slurries were blended in equal volumes to produce the finished slurry. Initial sample testing indicated that the viscosity was too great in Slurry 2, so it was diluted with water. The final mixture contained 8.2 per cent bentonite instead of 10 per cent. Future work was done using 8 per cent.

The two slurries were pumped with Moyno pumps. An equal volume was maintained from each pump. This was accomplished by prior pump tests to calibrate the pumps so the volume could be known by reading the pump speed. The volume pumped was indicated on special meters having dial faces reading in gallons per minute.

The length of pipe after the two slurries blended was 36 feet. The placement of the slurry was controlled by an operator moving the pipe to turn the nozzle from side to side. Figure 20 illustrates how the slurry was placed by oscillation of the pipe. Figure 21 shows the completed bulkhead which completely filled and sealed the simulated drift.

Following the successful completion of the bulkhead in the simulated mine drift, preparation was started to build a similar bulkhead in an abandoned mine drift. Mine No. 62-008 was selected for this field test.

Before any field tests could be made, it was necessary to secure access rights to the properties involved. Obtaining these access rights usually requires considerable time and money. Legal services are normally required. In some cases, a large number of owners are involved, each

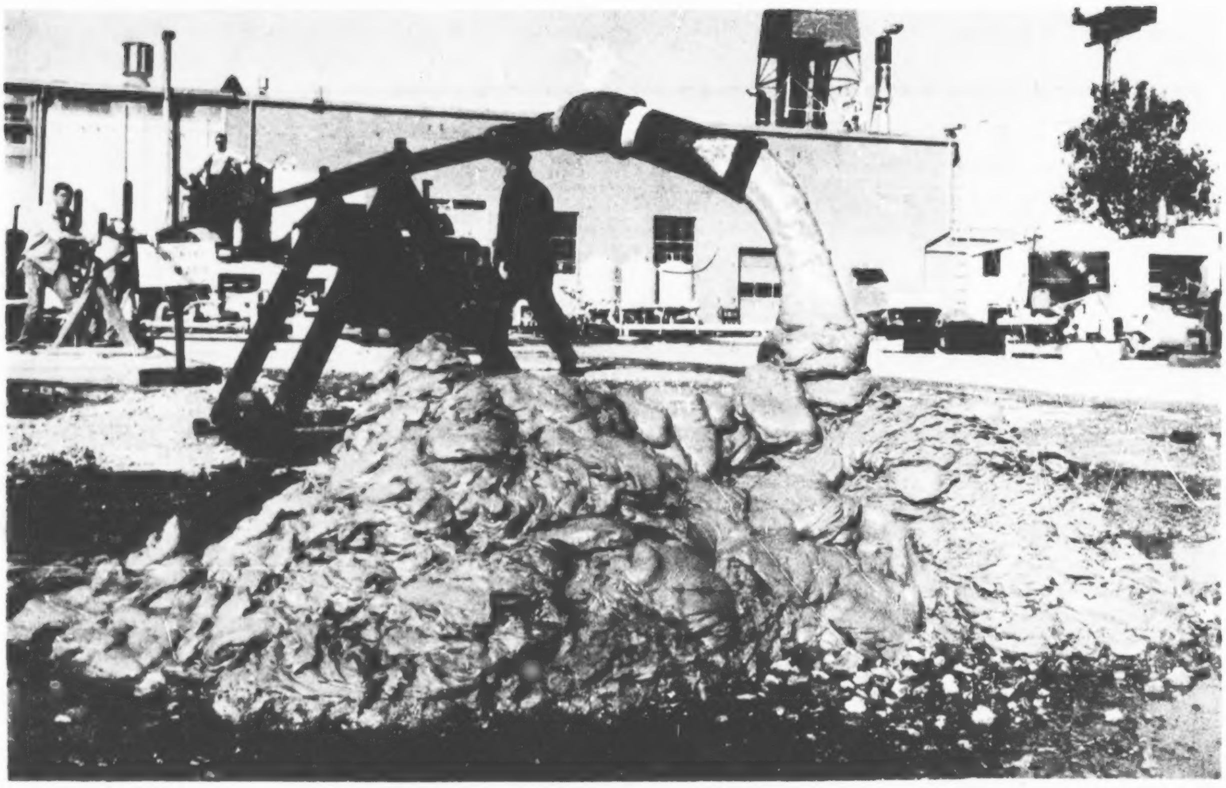


FIGURE 17 - BUILDUP OF GYPSUM CEMENT SLURRY IN YARD TEST

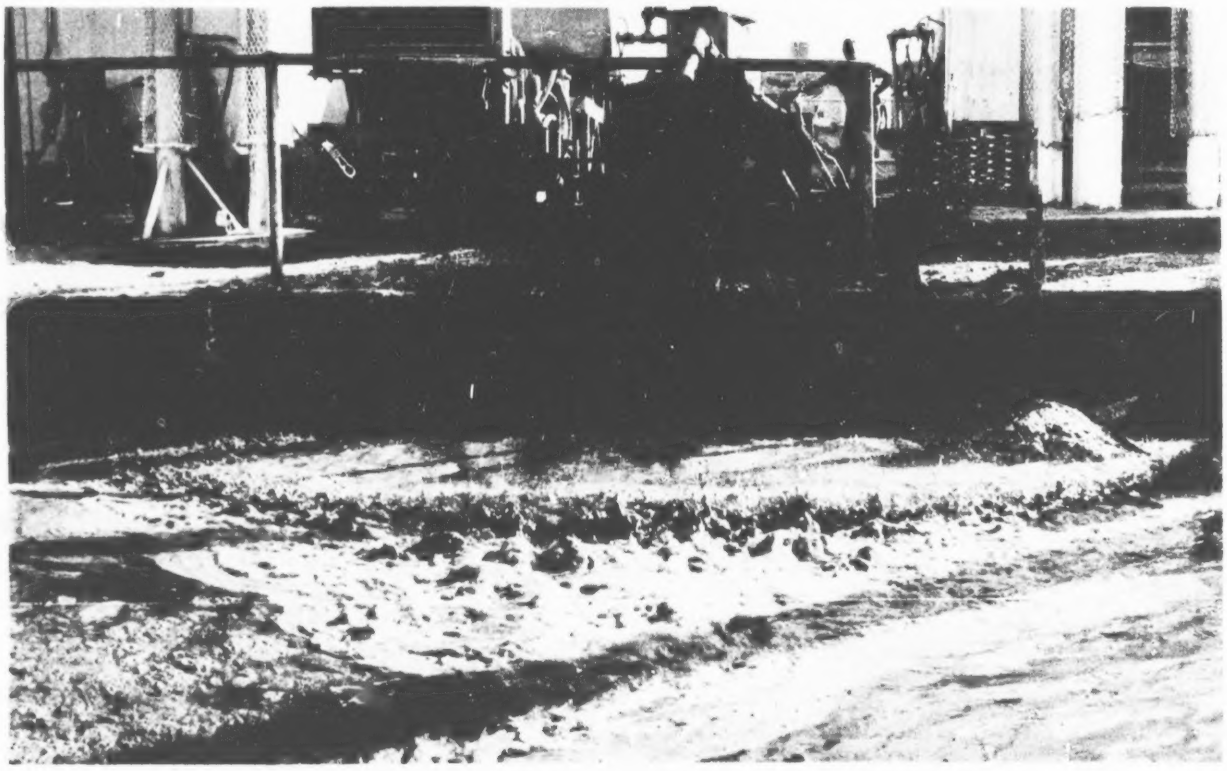


FIGURE 18 - NOZZLE DEVELOPMENT TEST

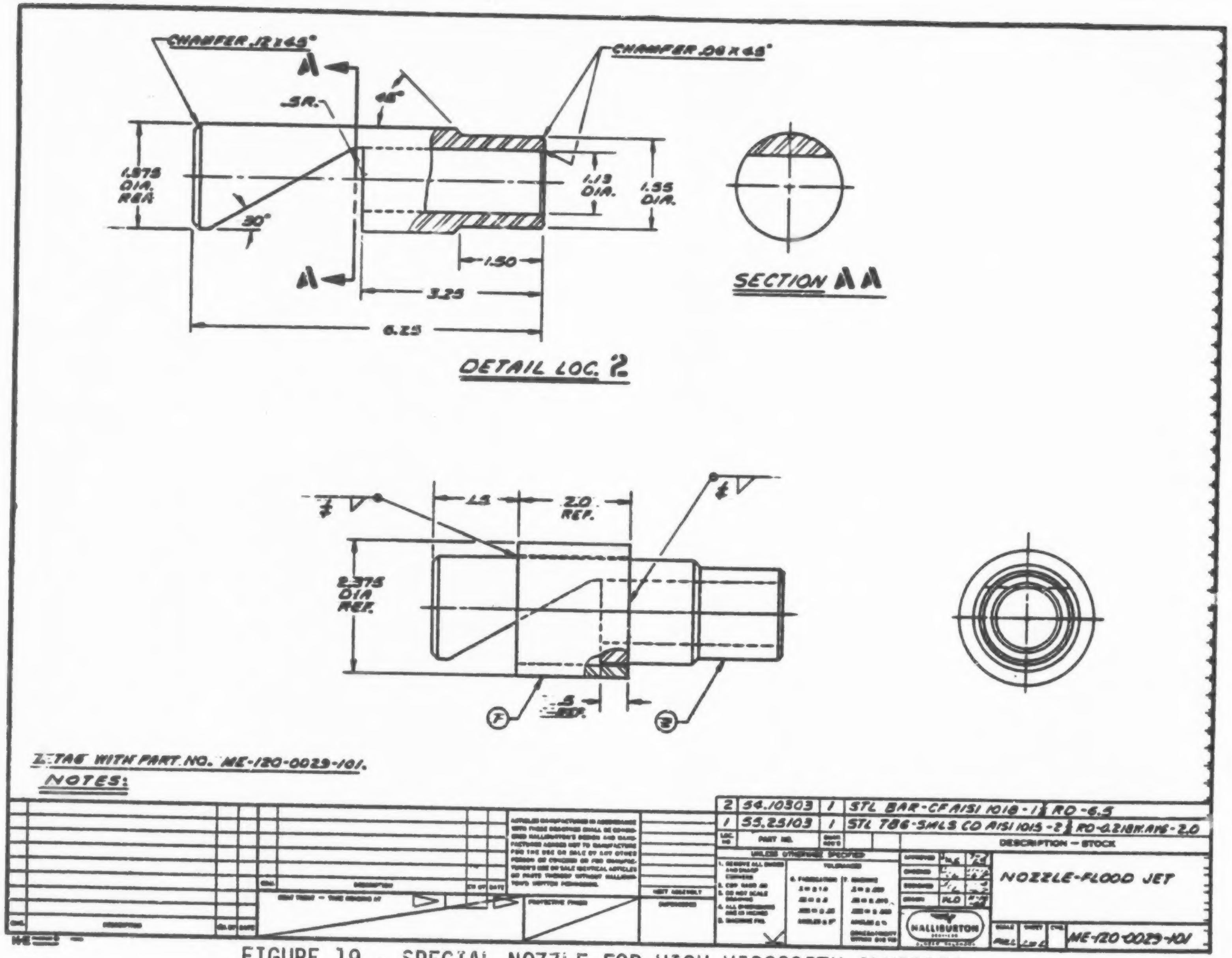

CHAMFER .12 x 45°
A
.SR.
45°
1.875 DIA. REF.
30°
A
1.50
3.25
6.25
CHAMFER .06 x 45°
1.13 DIA.
1.55 DIA.
SECTION A A
DETAIL LOC. 2
1.5
2.0 REF.
2.375 DIA REF.
.5 REF.
F
B
2 TAG WITH PART NO. ME-120-0029-101.
NOTES:
2 | 54.10303 | 1 | STL BAR-CF AISI 1018 - 1 RD - 6.5
1 | 55.25103 | 1 | STL TBG-SMLS CD AISI 1015 - 2 RD-0.218W.AWG-2.0
LOC. NO.
PART NO.
DESCRIPTION — STOCK
UNLESS OTHERWISE SPECIFIED
TOLERANCES
NOZZLE-FLOOD JET
PLO
HALLIBURTON SERVICES
FULL
ME-120-0029-101

FIGURE 19 - SPECIAL NOZZLE FOR HIGH VISCOSITY SLURRIES

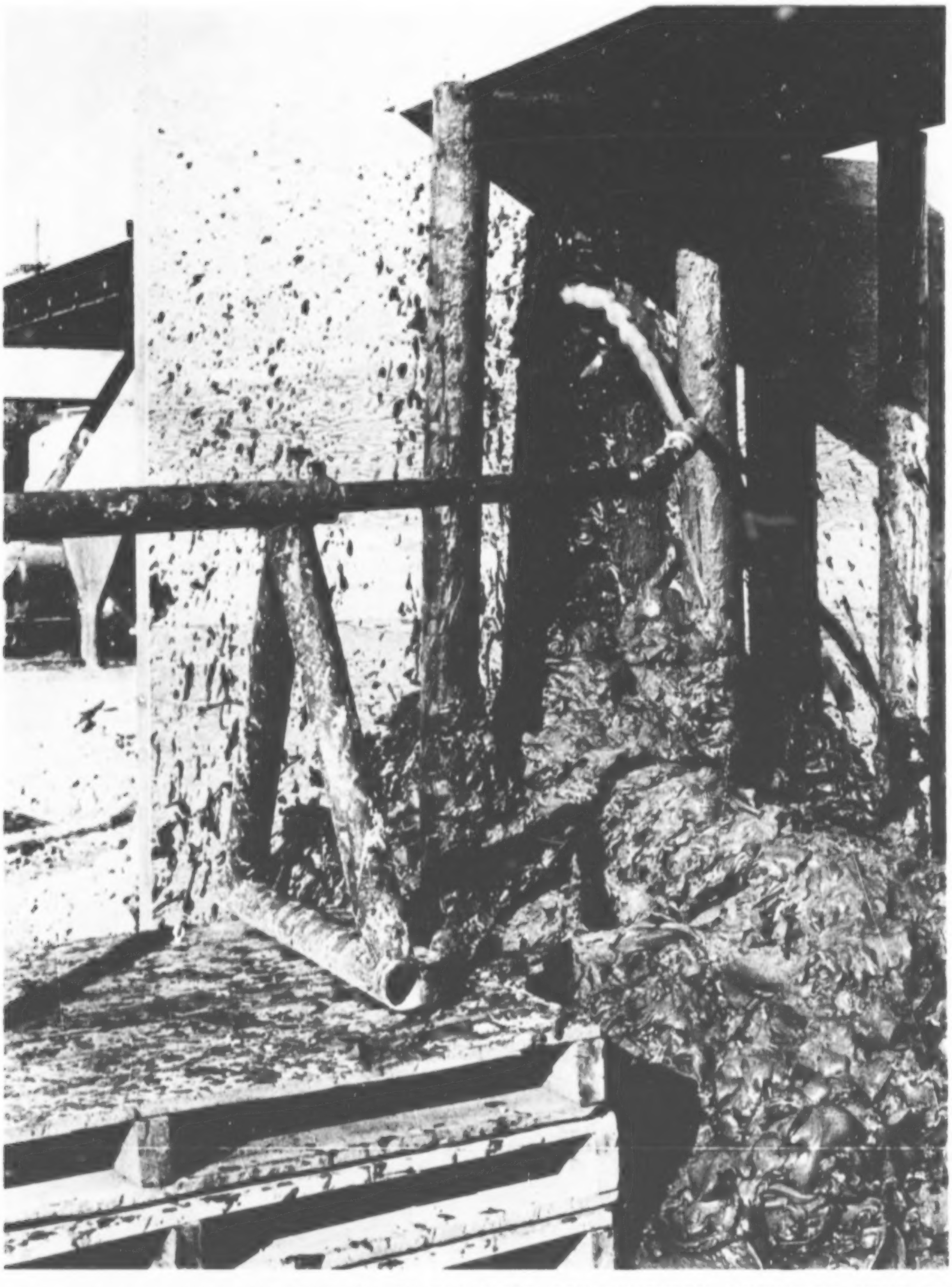


FIGURE 20-SLURRY DISTRIBUTION IN BULKHEAD CONSTRUCTION-SIMULATED MINE TEST

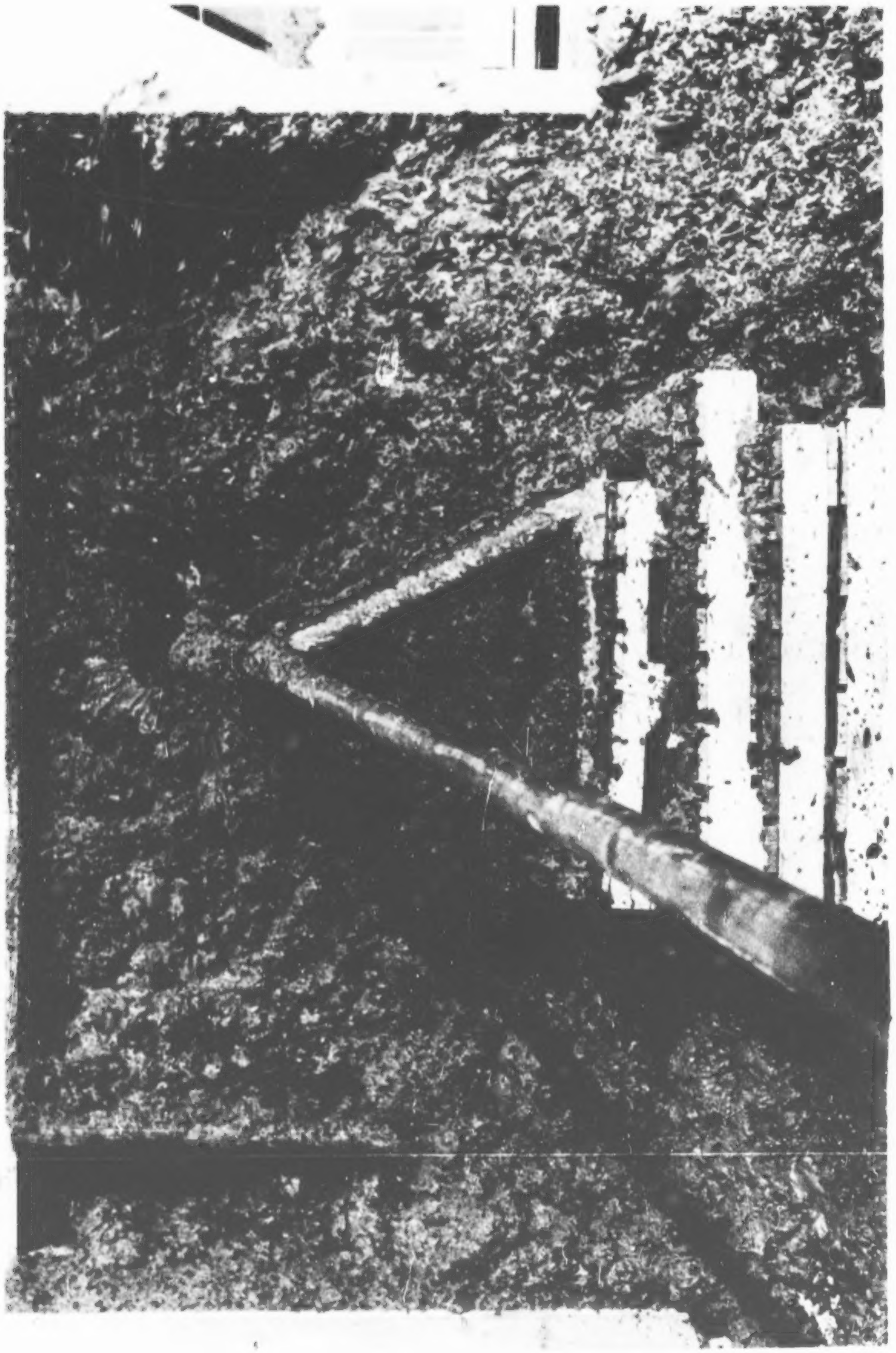


FIGURE 21 - COMPLETED BULKHEAD - SIMULATED MINE TEST

of whom must be located to complete the access agreement.

Mine No. 62-008 is a small abandoned drift mine complex located 3 miles west of Lost Creek, Harrison County, West Virginia. Twenty acres of Pittsburgh coal have been deep-mined and approximately 130 acres remain in reserve. The Redstone coal contains a small drift mine and has been augered along with the Pittsburgh seam. The periphery has been stripped for both the Pittsburgh and Redstone coal, leaving a steep continuous highwall.

A mine map of this mine is attached as Figure 22. Three openings are evident in the Redstone coal, but only seepage was observed from these openings. Four openings were observed in the Pittsburgh coal and three were draining slightly. The fourth opening was a 4-foot diameter airway which was partially collapsed and dry. The flow from these three openings was measured previously at 5 gpm. Sample analysis indicated a pH of 2.85 and an acidity of 459 mg/l, or 27.5 pounds per day of acid production. Analysis also indicated an iron content of 119 mg/l, or 7.1 pounds per day of iron, and 3500 mg/l of sulfate. This water is being discharged into a tributary of Lost Creek. The coal seams dip steeply to the east. Three fresh water impoundments were found on the site. A geologic section of this mine is attached (Figure 65 - Appendix).

The opening selected for the first field test was the main portal, denoted as Opening No. 4. A view of this opening is shown in Figure 23 before starting any cleanup or preparation. This opening was partially open and had good roof conditions. Figure 24 shows the openings in Mine No. 62-008 after site was cleaned, openings enlarged and excessive backfill removed. The main portal is the left opening in this picture. Drift is 5 feet high and 11 feet wide.

Figure 25 shows the inside of the main portal before cleanup. The portal included posts in the interior similar to those in the simulated mine test. After cleanup, the floor, walls and roof were jetted with water to remove mud and slime to provide a good bonding surface for the cement bulkhead. The jetting was accomplished by pumping with a high-pressure pump through the pipe and nozzle on the two-wheel dolly, which was later used to place the slurry in the mine for the bulkhead. A 2-inch plastic line was placed on the floor to extend from behind the bulkhead to serve as a drain line.

The two slurries were mixed in 250-cubic foot cone-bottom tanks. The bentonite was prehydrated by pumping water through the regular Halliburton mixing system and slowly adding the bentonite. After this was mixed and stirred by air for 30 minutes, sodium silicate was added through the same system and mixed with air in the bentonite slurry. The cement was mixed by blowing bulk cement from a pressurized tank into the tank of water and stirring with air jets. The temperature of the cement slurry was 67°F and the weight was 15.4 lbs/gal. The temperature of the gelled silicate slurry was 45°F.

These slurries were again batch-mixed, using the following proportions:

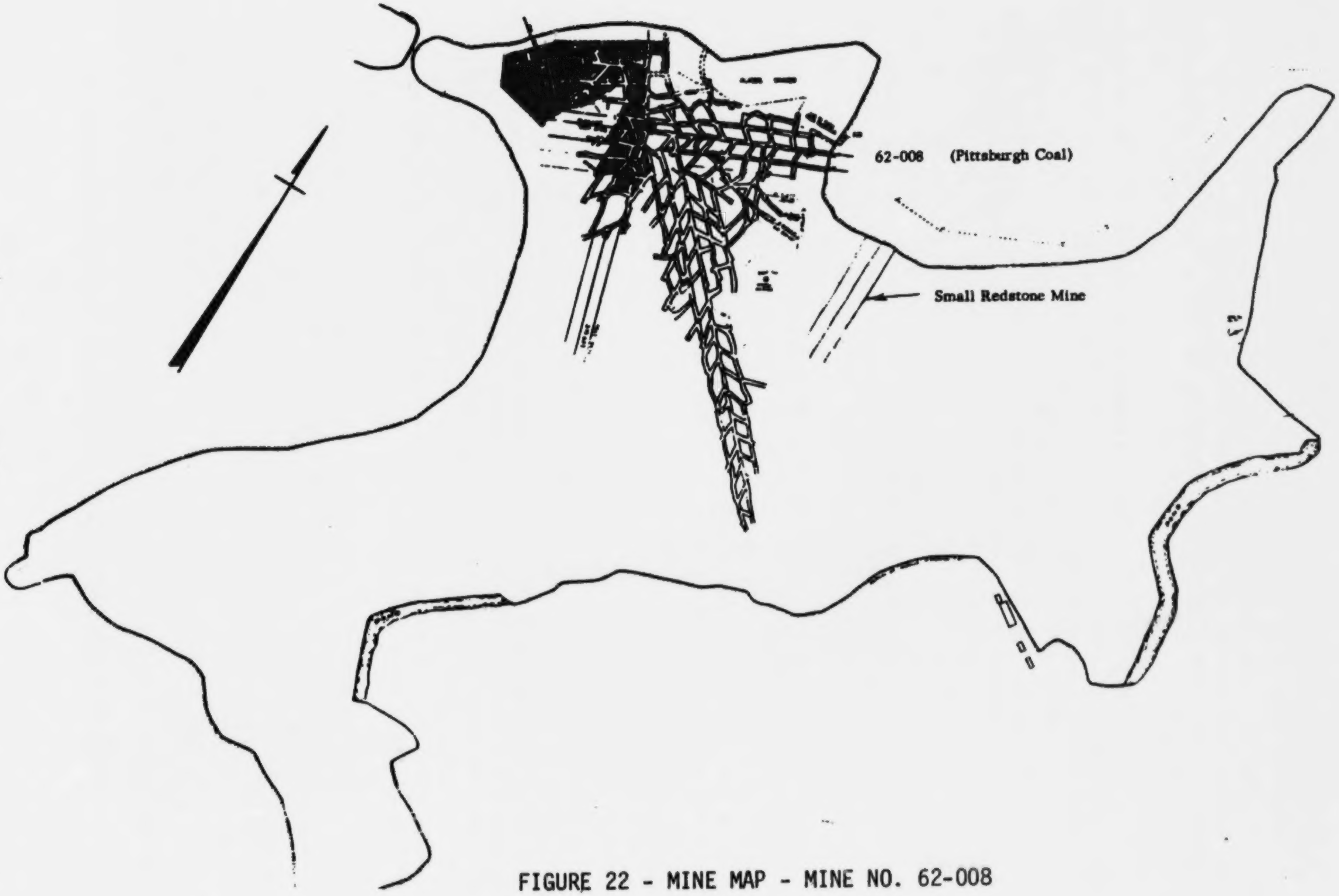

62-008 (Pittsburgh Coal)
Small Redstone Mine

FIGURE 22 - MINE MAP - MINE NO. 62-008

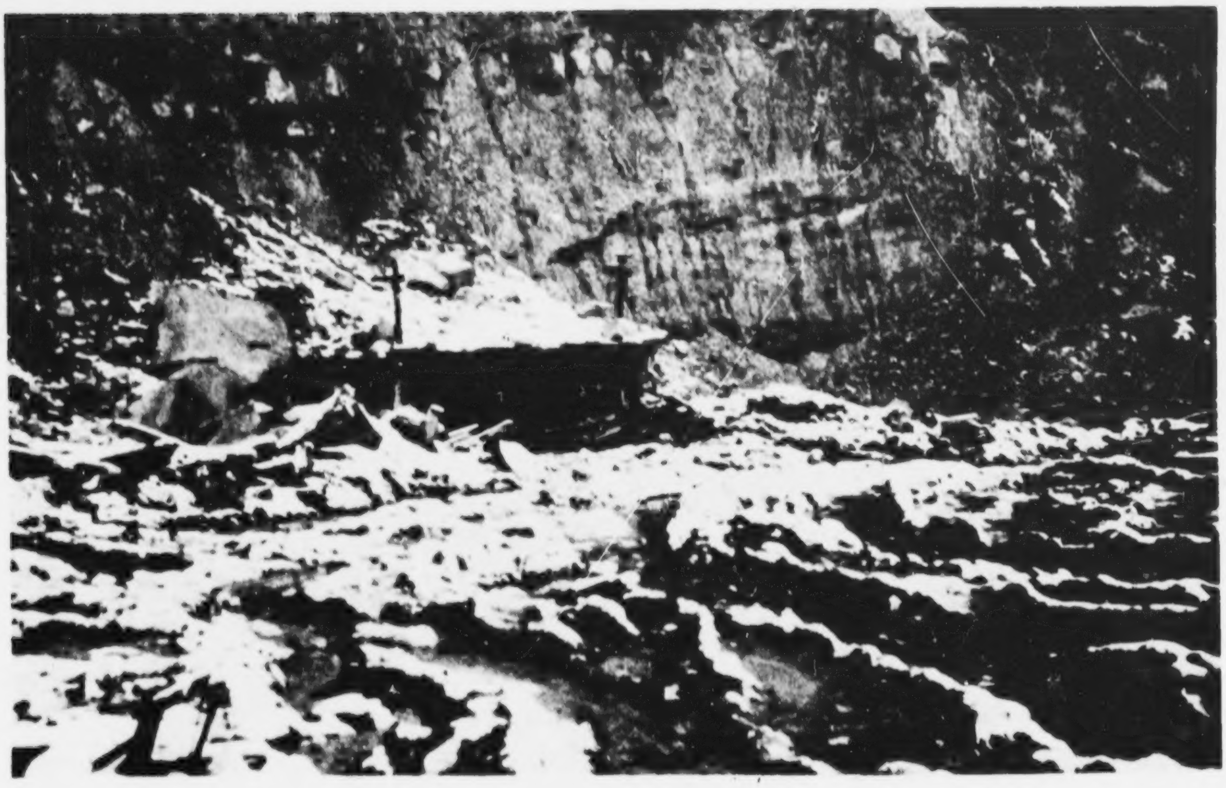


FIGURE 23 - MAIN PORTAL - MINE NO. 62-008 BEFORE CLEANUP

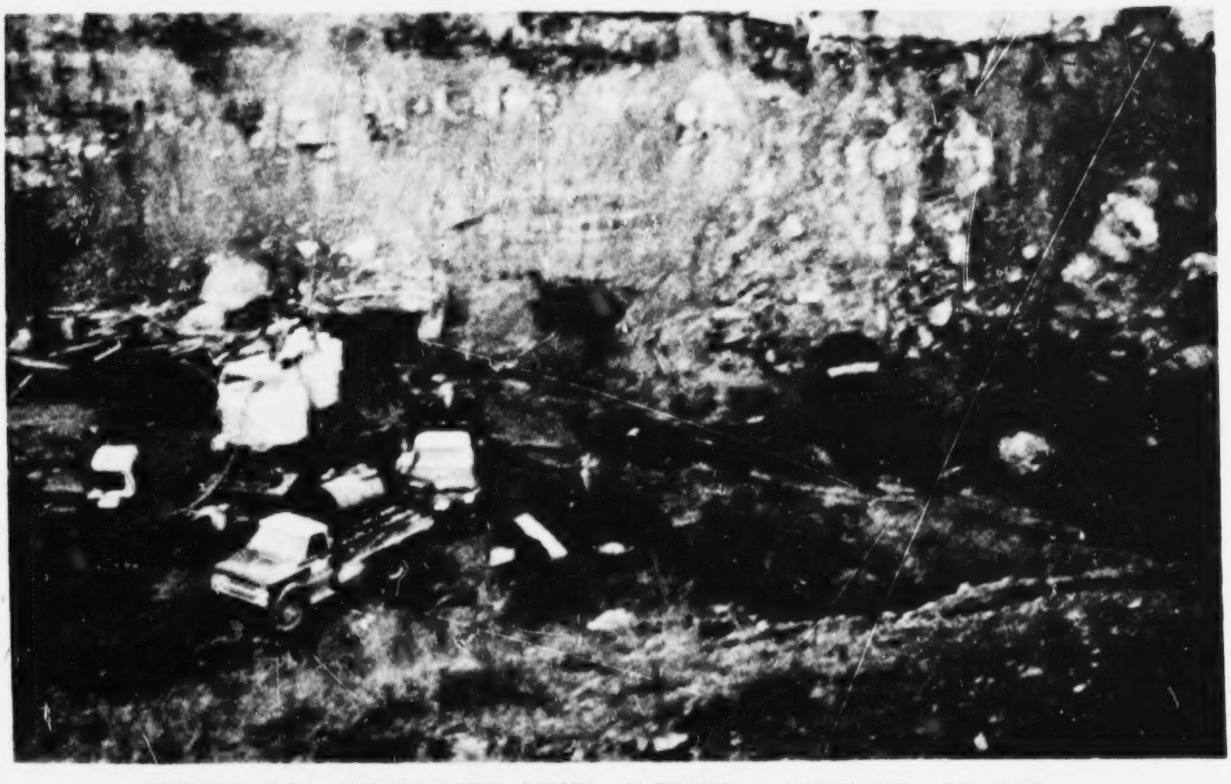


FIGURE 24 - WORK SITE AFTER CLEANUP - MINE NO. 62-008

FIGURE 25 - INSIDE OF MAIN PORTAL BEFORE CLEANUP - MINE NO. 62-008

Slurry 1	
Water	816 gallons
Cement	168 sacks

Slurry 2	
Water	1,050 gallons
Bentonite	700 pounds
Sodium Silicate	350 gallons

Sample tests mixing the two slurries showed that the resultant slurry had sufficient viscosity, but would not set quickly. It was thought that the cause for this was the low temperature of the slurries, particularly the bentonite-silicate slurry. To offset this, additional sodium silicate was added for the sample tests. This proved successful, so an additional 165 gallons of sodium silicate was added to Slurry 2. These slurries were then used to blend together for the bulkhead construction.

The building of the bulkhead was started by pumping equal volumes of each slurry, bringing them together in a manifold just outside the mine. The blended slurry was then pumped through a perforated plate in the discharge pipe and on into the drift. The beginning of the bulkhead construction is shown in Figure 26. The completed bulkhead in this drift is shown in Figure 27. Fifty minutes were required to build the bulkhead after starting to pump the two slurries into the mine drift. The angle of repose on the front side of the bulkhead was about 60 degrees from the horizontal. A drawing of a section through this bulkhead is attached as Figure 28.

It was discovered during the two full-scale tests that there were several critical points. The temperatures of the slurries for the mine bulkhead were low, causing a problem in getting the proper setting time. The technique of proper placement to prevent sloughing was found to be very important, as well as maintaining the proper pump rates and slurry ratio.

Because of the low temperature problem, tests were conducted in the laboratory using materials at 40°F. Slurries were mixed using bentonite-sodium silicate ratios of 2 to 1 and 3 to 1. Bentonite ranged from 8 per cent to 12 per cent. Some tests included small amounts of sodium bicarbonate. It was found that when the temperature was low, especially below 50°F, it had an influence on the setting properties of the final slurry, regardless of the amount of bentonite used or the degree of its hydration. The effect was more pronounced as the per cent of bentonite was reduced. The data indicates that a bentonite-sodium silicate ratio of 2 to 1 rather than 3 to 1 was necessary when the temperature was 40°F or 50°F (Table 15 - Appendix).

Full-scale tests were again conducted in the yard by spraying slurry on the ground for observation. The effect of using a new mixing tube to replace the perforated disc was evaluated and found to provide excellent mixing with considerable reduction in pressure requirement. This mixer nipple is an 18-inch long section of 1-1/2-inch pipe with alternate horizontal and vertical thin section baffles. These act as shear plates to impart shear to the fluid which acts to mix or blend all parts of the slurry. A drawing of this mixer nipple is shown in Figure 29. This type mixer nipple was used in the discharge line

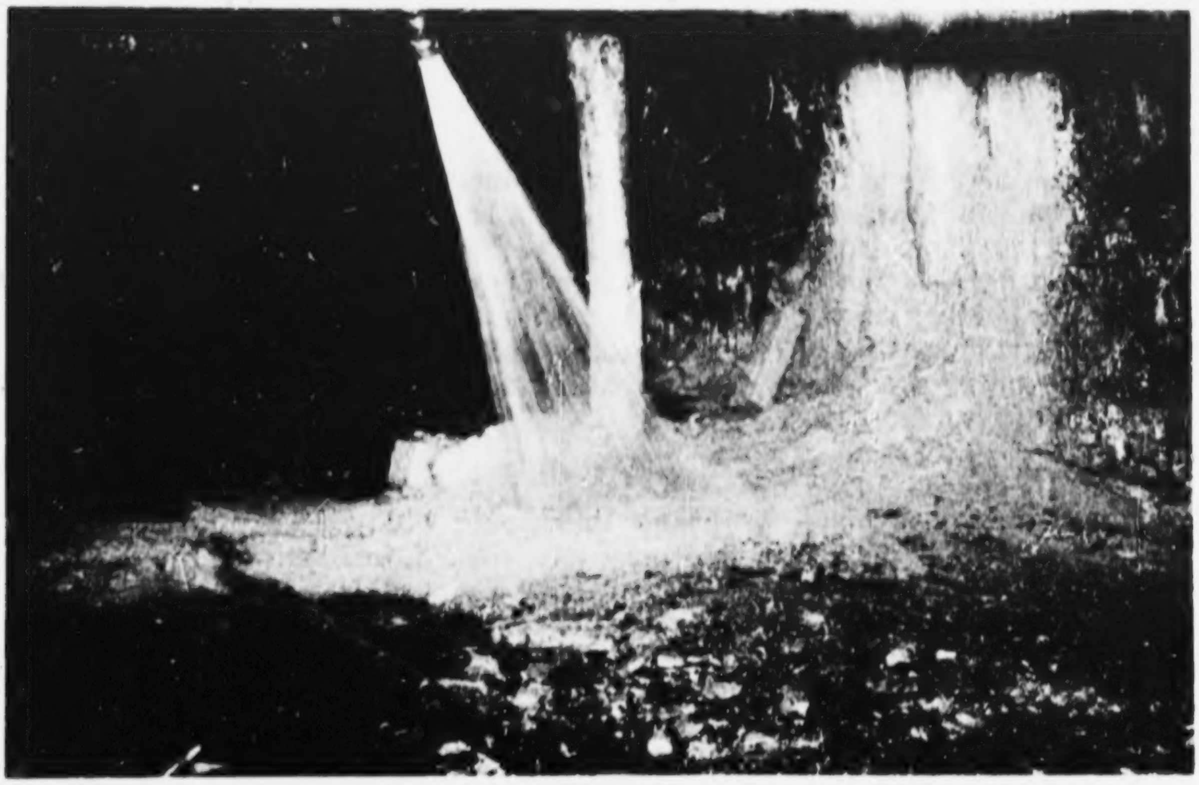


FIGURE 26 - BEGINNING OF BULKHEAD CONSTRUCTION IN MINE NO. 62-008

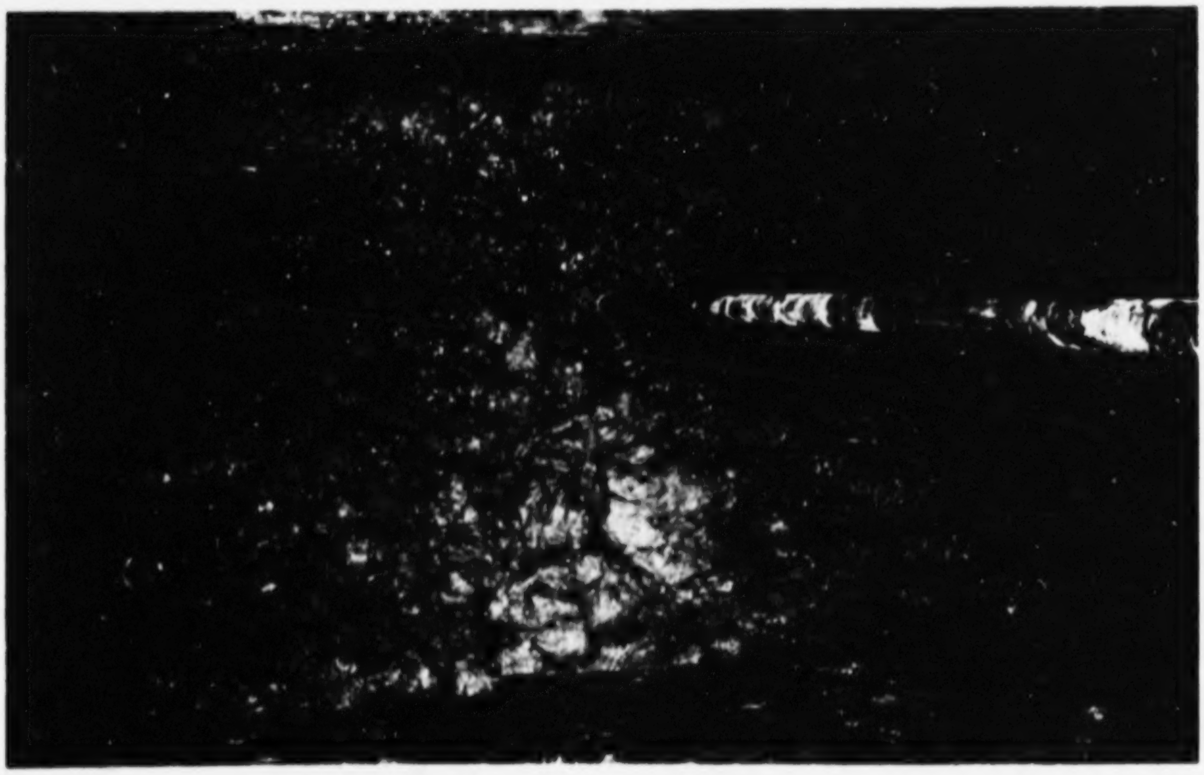
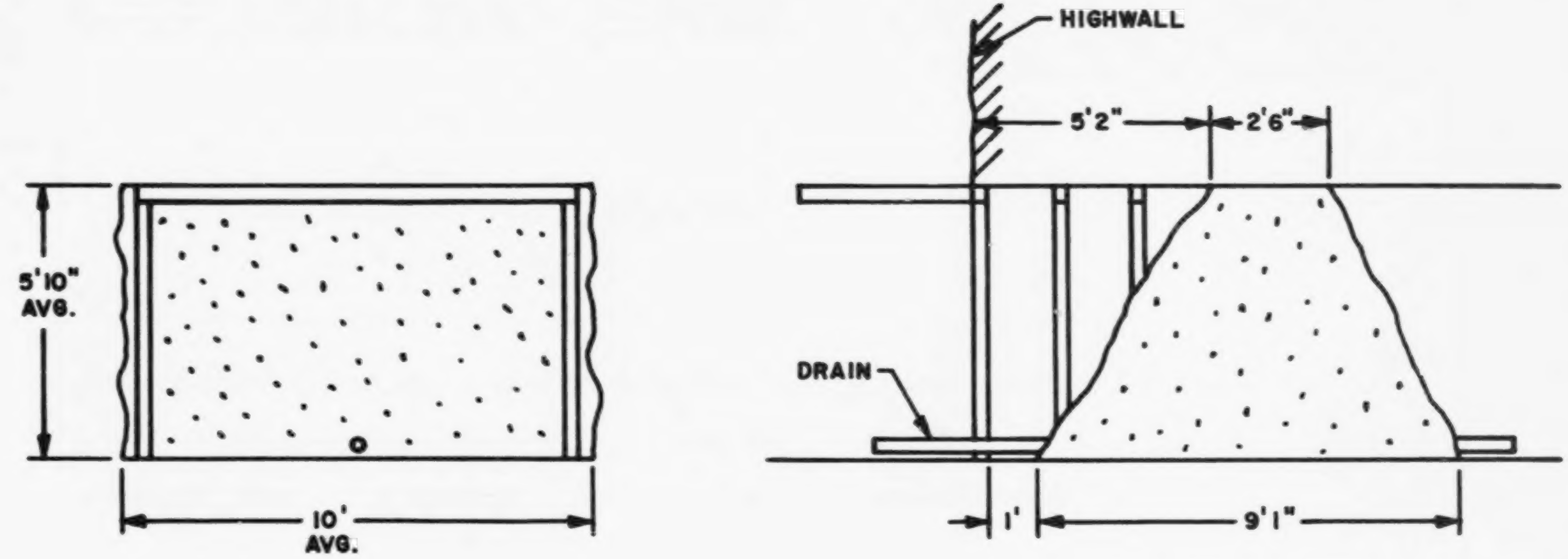


FIGURE 27 - COMPLETED BULKHEAD IN MINE NO. 62-008


5'10"
AVG.
10'
AVG.
HIGHWALL
5'2"
2'6"
DRAIN
1'
9'1"

SECTION - QUICK SETTING BULKHEAD - OPENING NO. 4 - MINE NO. 62-008

FIGURE 28

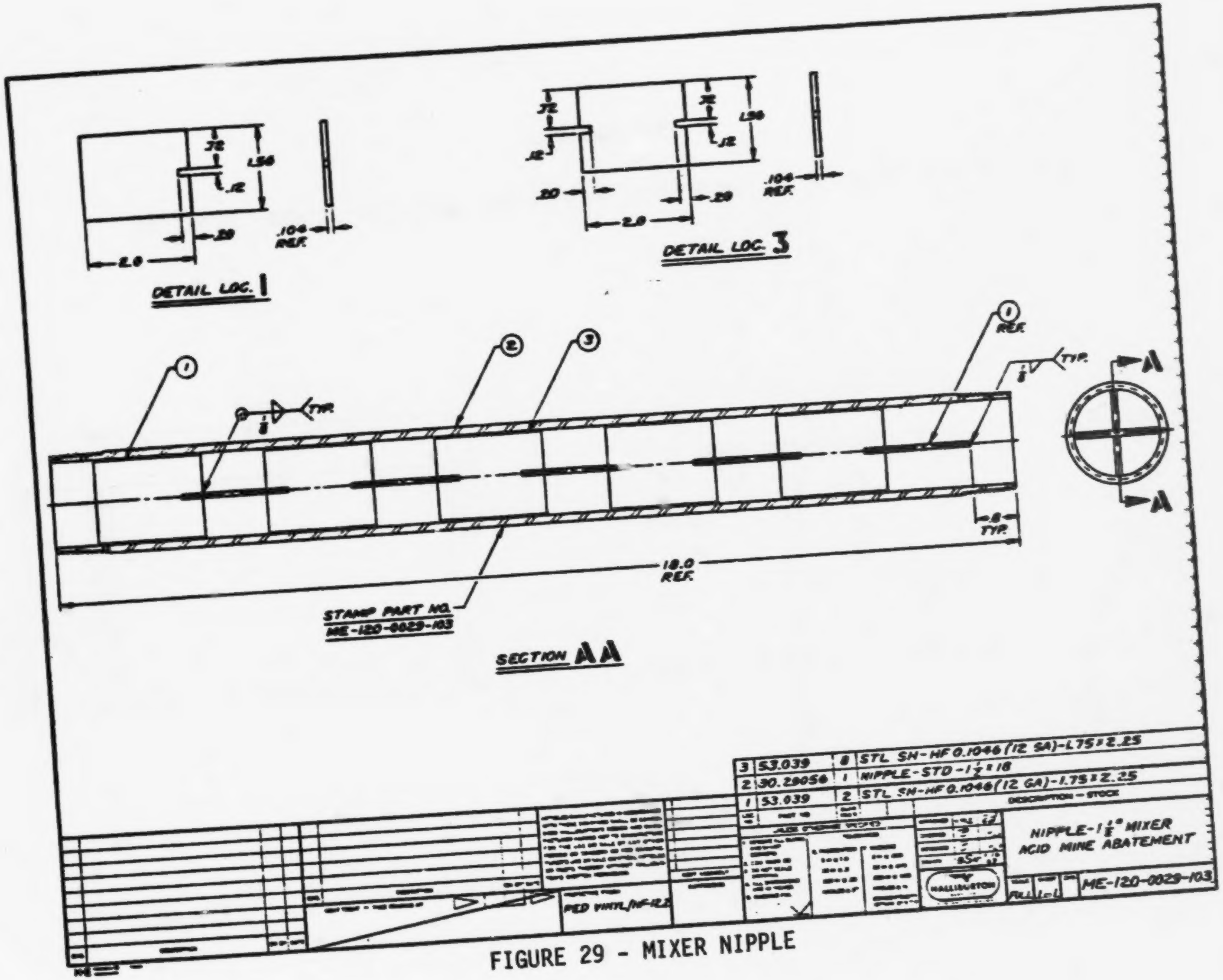

.72
1.56
.12
.20
2.0
.104 REF.
DETAIL LOC. 1
.72
.12
1.56
.20
2.0
.104 REF.
DETAIL LOC. 3
①
②
③
① REF.
1/8 TYP.
1/8 TYP.
A
A
.8 TYP.
18.0 REF.
STAMP PART NO. ME-120-0029-103
SECTION AA
3 | 53.039 | 8 | STL SH-HF 0.1046 (12 GA)-1.75 x 2.25
2 | 30.29056 | 1 | NIPPLE-STD-1½ x 18
1 | 53.039 | 2 | STL SH-HF 0.1046 (12 GA)-1.75 x 2.25
DESCRIPTION — STOCK
NIPPLE-1½" MIXER
ACID MINE ABATEMENT
HALLIBURTON
FULL
1 of 1
ME-120-0029-103
RED VINYL/HF-12.5

FIGURE 29 - MIXER NIPPLE

on all further testing.

The full-scale tests also confirmed the laboratory tests in regard to temperature. Slurries tested at low temperatures did not set quickly. The slurry would spread out before a set would occur. These tests also confirmed that the temperature should be at least 65°F in order to enable the slurry to set quickly and build up properly. Tests were also made by mixing the slurries with heated water. Water was heated to 78°F prior to mixing with cement or bentonite Using the original ratios of bentonite slurry to sodium silicate of 3 to 1, a slurry was produced which set quickly and built up into a wall with a high angle of repose. Figure 30 shows a view of this test. Two mixing nipples, providing 3 feet of length, were used on these and subsequent tests. The length of pipe used in the discharge line after the two slurries blended did not seem to be very critical. Lengths of 8 to 20 feet were used in most tests, but variations up to 30 feet did not change results appreciably. The controlling factor used in the tests which seemed most effective was the variation in pump rate and in the ratio of the two slurries. These were used as control methods when building bulkheads.

The yard tests indicated that there was little advantage in using 10 per cent bentonite if the water temperature was 65°F or higher, so 8 per cent was used as standard for further mixing.

Laboratory tests were made on slurry composed of 50% cement and 50% fly ash with calcium chloride as the accelerator. This material would not be less expensive than the silicate cement, but the mixing procedures would be simplified and therefore more economical. The tests showed that the slurry might be able to satisfactorily build a bulkhead, since the set time could be designed as low as 1-1/2 minutes with another minute required for self-support. However, full-scale yard tests proved that neither the set time nor the self-support time was short enough to permit the material to build up into a bulkhead, so this approach was abandoned.

Further field testing of the bulkhead construction materials and technique was scheduled for a drift opening in Mine No. 62-008 near Clarksburg, West Virginia. This portal, adjacent to the main opening, was designated as Opening No. 5. It was planned to build a seal in this opening by placing a front and rear bulkhead of quick-setting material and filling between the bulkheads with an expansive grout slurry. Flow of water from the drift prior to this test was 15 gpm. An analysis indicated that the discharge had a pH of 2.8, acidity of 2260 mg/l and iron content of 600 mg/l. This amounts to a daily production of 413 pounds of acid and 100 pounds of iron.

The equipment used for placing this seal in the mine was the same as used previously to build the bulkhead in an adjoining opening (see Figure 24), with the addition of a large tank which was used for

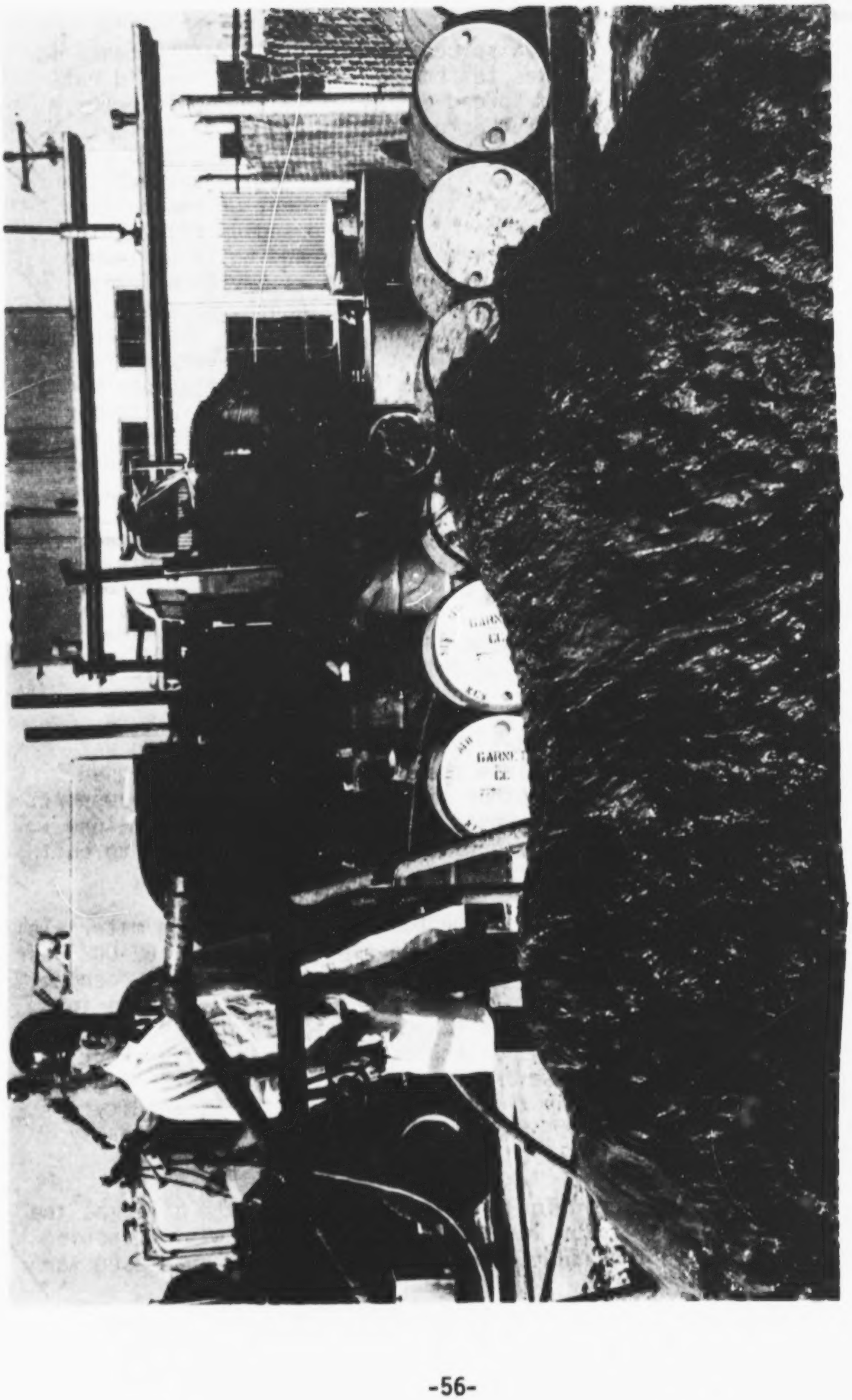


FIGURE 30 - YARD TESTS OF SLURRY USING HEATED WATER

heating the water. The work was conducted when the temperature was about 21°F and snowing. Water was heated to 62°F to mix the slurries. Cement used was API Class A. Eight per cent bentonite was used with a ratio of 3 to 1 for the prehydrated bentonite-sodium silicate slurry. A volume of 12 cubic yards was prepared for the rear bulkhead.

The dolly which was used to carry the discharge pipe and slurry lines permitted the operator to work remotely from the place where bulkhead was built. Figure 31 shows this equipment in use on the mine seal construction. There were 17 feet of discharge pipe ahead of the swivel where the two slurries blended, including the two mixer nipples. The dolly contained two electric lights mounted just behind the nozzle for illuminating the discharge area.

When the operation of placing the slurry began, the heat from the slurry caused a fog around the bulkhead. This made it difficult for the operator to see the progress of work, but did not cause any insurmountable problem. The construction of the bulkhead was complete in less than one hour and required 12 cubic yards of slurry.

After this bulkhead was completed, 2-inch plastic pipes were installed at floor and roof from the area in front of the bulkhead to the outside entrance and fitted with valves to control the grouting operation. These were used for grouting the interior part after constructing the front bulkhead. Figure 32 is a schematic drawing showing grout pipe locations.

The front bulkhead was built in the same manner as the rear bulkhead. Water temperatures were 59°F for the bentonite slurry and 62°F for the cement slurry. The ambient temperature was 20°F. Since the temperatures of the slurries were slightly below the desired minimum, special care was given to material placement, slurry ratios and pump rates to assure the material setting sufficiently to support its weight. The construction of this bulkhead was completed in slightly over one hour after materials were mixed and required 10 cubic yards of slurry.

On the following day, water containing a dye was pumped into the void between the two bulkheads to determine its volume and spot any leaks. When 4,410 gallons had been pumped into the void, a leak was observed around the upper left grout pipe. With a volume of 5,040 gallons, another leak occurred around the upper center grout pipe. Pumping was discontinued. The volume of the void was determined to be 773 cubic feet, based on volume of water required to fill to level of grout pipes and calculated volume above that point.

The void was filled by pumping Halliburton LIGHT Cement through the bottom grout pipe. This grouting material was a blend of 65% portland cement, 35% fly ash and 6% bentonite with 9.9 gallons of water per bulk cubic foot of dry material, or 61 parts cement, 26 parts fly ash, 5.2 parts bentonite and 82.5 parts water by weight.

When 478 cubic feet were injected, the grout flowed from the upper left grout pipe. Valve was closed and pumping continued until 773 cubic feet were injected. At this time, leakage was observed across a large portion of the top of the bulkhead. The cement slurry was drained until the level was below the upper grout pipes and then allowed to harden.

The second stage of grouting was accomplished by pumping the grout through the upper right grout pipe. The same grout slurry was used as in the first stage with the addition of sawdust, shredded cellophane and shredded cane fiber. This grout containing the additives successfully sealed the leaks. A 4-foot head was placed on the slurry while it hardened by a riser pipe attached to the front of the center upper grout pipe. The drain line was fitted with a manometer to measure the head of water which would form behind the mine seal. A drawing of a section through this mine seal is attached as Figure 33.

The three bulkheads built of the sodium silicate, bentonite and cement mixture have proven to be excellent seals. A schematic drawing showing these seals in place in Mine No. 62-008 is attached as Figure 34.

Monitoring of the water head retained by the seals was begun immediately after construction and continued throughout the project. After 10 months, the single bulkhead in Opening No. 4 showed only slight seepage and contained a head of 54.5 inches behind it. The double bulkhead seal in Opening No. 5 after 7 months showed no leakage and contained a head of 51 inches of water (Tables 16 and 17). There were no water samples collected from Openings 4 and 5 after installation of seals because the drainage stopped.

The cost for the single bulkhead in Opening No. 4 was $3,564.00. This included $647.00 for site preparation, $1,165.00 for materials and $1,752.00 for equipment and operators. The construction cost for the double bulkhead seal with filler material was $9,449.00. This included $894.00 for site preparation, $3,872.00 for materials and $4,683.00 for equipment and operators. This cost could probably be reduced when this type of construction is performed on an operational basis.

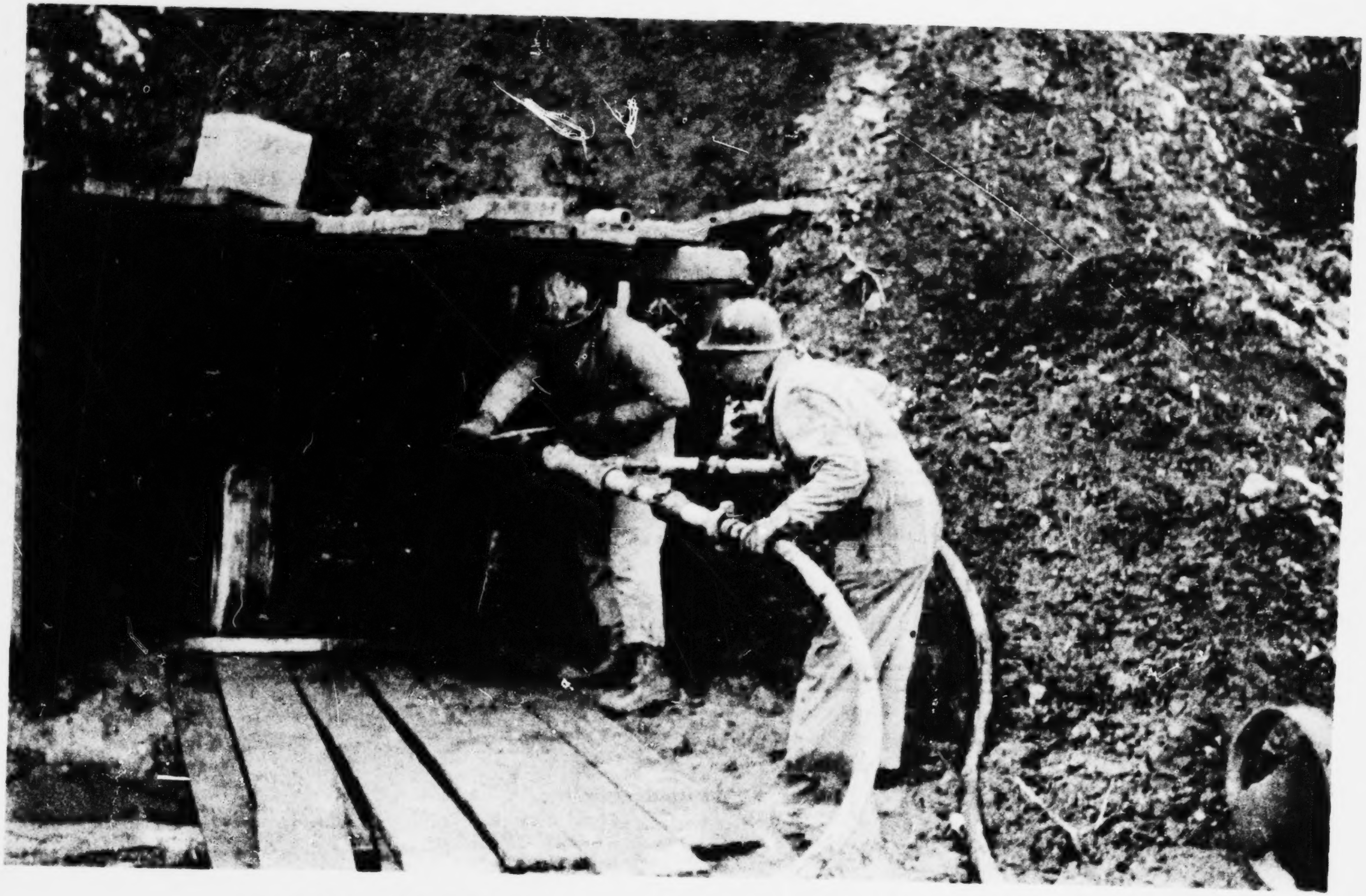
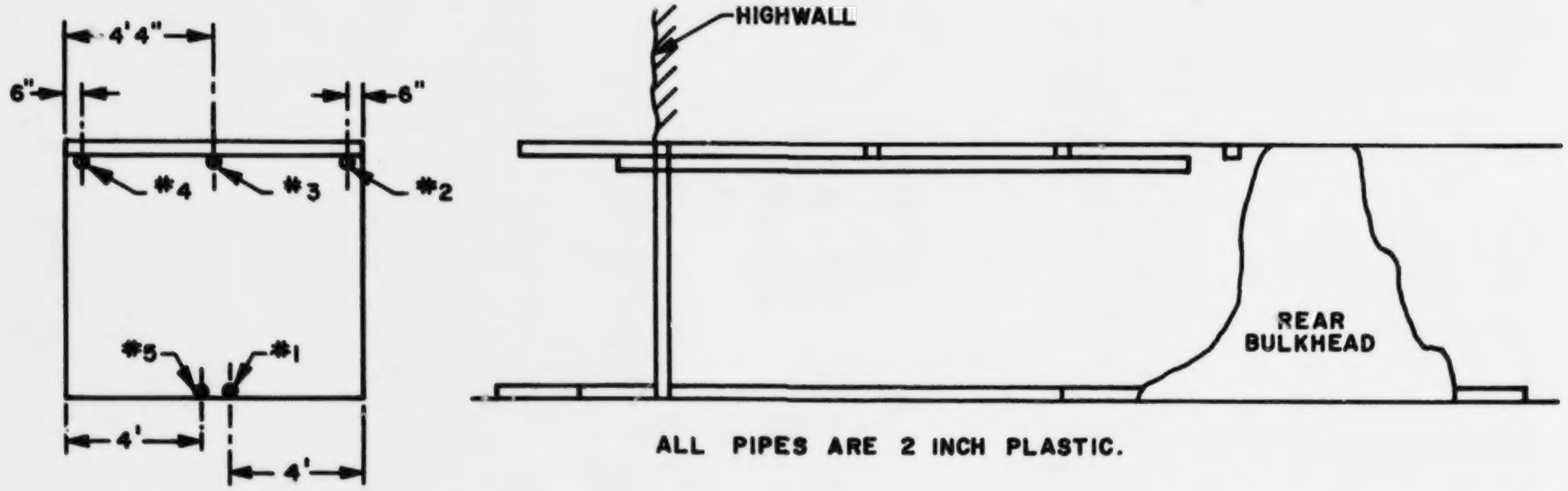


FIGURE 31 - REMOTE OPERATION OF EQUIPMENT FOR PLACEMENT OF SLURRY

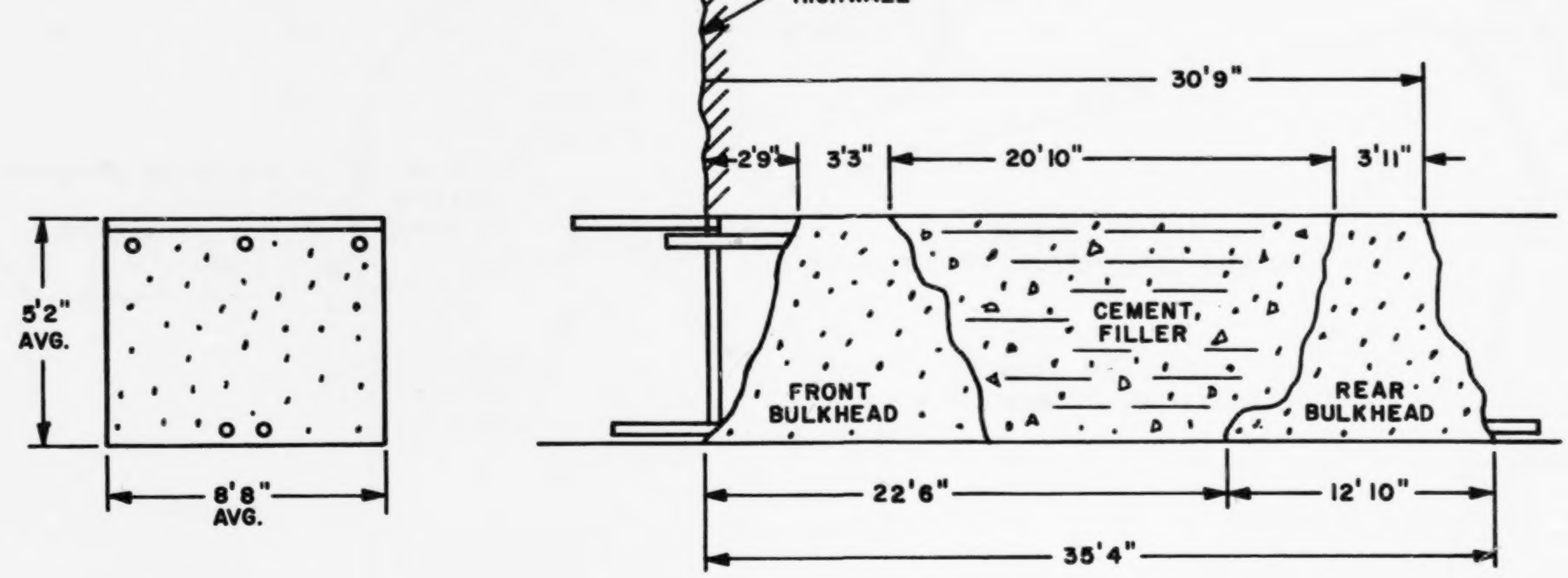

4'4"
6"
6"
#4
#3
#2
#5
#1
4'
4'
HIGHWALL
REAR BULKHEAD

ALL PIPES ARE 2 INCH PLASTIC.

#1 - LENGTH 19 FEET
#2, #4 - LENGTH 12.5 FEET
#3 - LENGTH 20 FEET
#5 - DRAIN PIPE - LENGTH 40 FEET

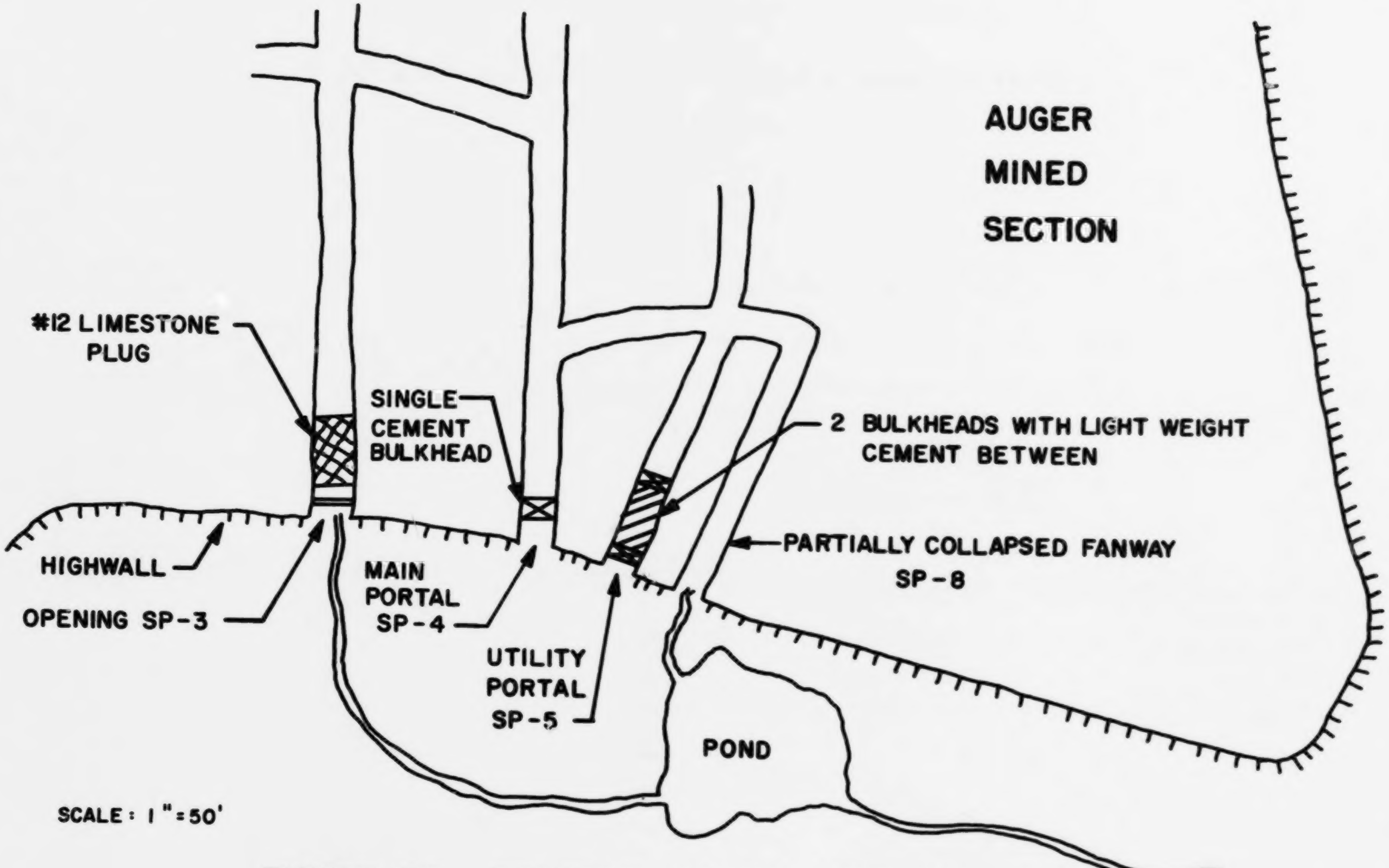
GROUT PIPE LOCATION - OPENING NO. 5 - MINE NO. 62-008

FIGURE 32


HIGHWALL
30'9"
2'9"
3'3"
20'10"
3'11"
CEMENT, FILLER
FRONT BULKHEAD
REAR BULKHEAD
22'6"
12'10"
35'4"
5'2" AVG.
8'8" AVG.

SECTION - MINE SEAL - OPENING NO. 5 - MINE NO. 62-008

FIGURE 33



AUGER
MINED
SECTION
#12 LIMESTONE PLUG
SINGLE CEMENT BULKHEAD
2 BULKHEADS WITH LIGHT WEIGHT CEMENT BETWEEN
PARTIALLY COLLAPSED FANWAY SP-8
HIGHWALL
OPENING SP-3
MAIN PORTAL SP-4
UTILITY PORTAL SP-5
POND
SCALE: 1"=50'

FIGURE 34 - REMEDIAL WORK - MINE NO. 62-008

PERMEABLE PLUG INVESTIGATION

Research was conducted to determine the feasibility of using permeable plugs to provide for drainage of some mine water with in-place treatment of the fluid as it passes through the plug.

Several approaches were investigated. One possibility was to use some type of mine seal containing a conduction pipe. The pipe was to contain either an electrode to remove the iron or a bed of ion exchange resin which could be recharged to remove sulfate and iron. Another proposal was made to use calcium phosphate, barium carbonate or phosphate rock in a drainage pipe to treat the water. In this method, as materials were expended they could be replaced with fresh chemicals. These methods were rejected from further consideration because of the maintenance problem involved.

The idea which seemed to have the most merit was that of pneumatically placing a plug of graded aggregate in a mine drift. The aggregate would be so graded that acid mine water would flow through the plug with sufficient retention time to be neutralized. As iron hydroxide (and possibly calcium sulfate) resulted from the neutralization process, the pores would gradually fill so that the result would be a plug which would seal the mine drift.

Two different approaches were taken. The first idea was to use an aggregate which would not seal quickly but would neutralize the water flowing through the plug. The other approach was to use an aggregate which would quickly form a seal.

Various grades of aggregate were investigated to find the proper size distribution which might be adaptable to one of the desired approaches. Screen analyses were made of samples from two vendors in the project area. These vendors were Paul Harrold Limestone Company, Clarksburg, West Virginia and Greer Limestone Company of Greer, West Virginia. The screen analyses are attached in Table 18 of the Appendix.

The permeability of a plug composed of graded limestone submitted by each company was determined. Permeability is a measurement of the resistance to fluid flow offered by a known volume of particles, usually expressed in darcies. Darcy's law, which states that the velocity of flow of a liquid through a porous medium due to difference in pressure is proportional to the pressure gradient in the direction of flow, permits the determination of permeability mathematically. Flow rate was determined while maintaining a constant pressure drop under controlled laboratory test conditions through a definite size sample of graded limestone using water as the fluid. These values were then used in a computer program to calculate the permeability using the following formula based on Darcy's law:

$$K = \frac{\mu \; L \; Q}{A \; \Delta P} \qquad (I)$$

where K = Permeability, Darcies

μ = Viscosity, centipoise

Q = Flow Rate, ml/sec

L = Length over which pressure drop measured, cm

ΔP = Pressure Drop, atmosphere

A = Cross-sectional area, sq cm

The permeability of the various limestone samples was determined to be as follows:

Aggregate	Permeability (Darcies)
Harrold #13	0.031
Greer #13	0.648
Harrold #12	1685.
Greer #12	4937.
Greer #12B	0.748
Harrold #10	8430.
Harrold #7	9535.

After a study of the screen analyses and permeability values, Harrold #12 limestone was selected for laboratory tests for a plug which would not seal quickly. Greer #12 and Harrold #10 and #7 had permeability so high it was felt that retention time involved would necessitate a plug too large to be practical.

The Greer #13 was modified by removing all particles that passed a 40 mesh (U.S. Standard Sieve) screen. This removed about 25% of the material in the sample. The modified Greer #13 material was then found to have a permeability of 210 Darcies. This material was then used for laboratory tests of the material to plug quickly. This material was also tested with 1% barium carbonate added to evaluate the effect of reduced permeability and to attempt to facilitate the precipitation of barium sulfate.

Formula I was used to calculate the length required for a limestone plug having a cross section of 5 x 10 feet, when the pressure drop was 2 psi and the flow rate 30 gpm. Assuming that the flow would only go through one foot of depth initially, it was found that a plug with a minimum length of 20 feet would be satisfactory for the Harrold #12 limestone.

The retention time for the acid mine water in the limestone bed was calculated using the following formula:

$$\text{Retention Time, minutes} = \frac{V\,\Phi}{Q} \qquad \text{(II)}$$

where V = Volume, cubic feet

Φ = Porosity, per cent

Q = Flow Rate, cfm

The minimum length of 20 feet was assumed for the 5 x 10 foot limestone plug. The porosity of the sample of Harrold #12 limestone was determined in the laboratory to be 40%. The flow was established at 30 gpm. The calculations showed the retention time for these conditions to be 23 minutes.

A laboratory model was constructed using a 6-inch column of limestone in a 1-inch glass tube. This apparatus is shown in Figure 35.

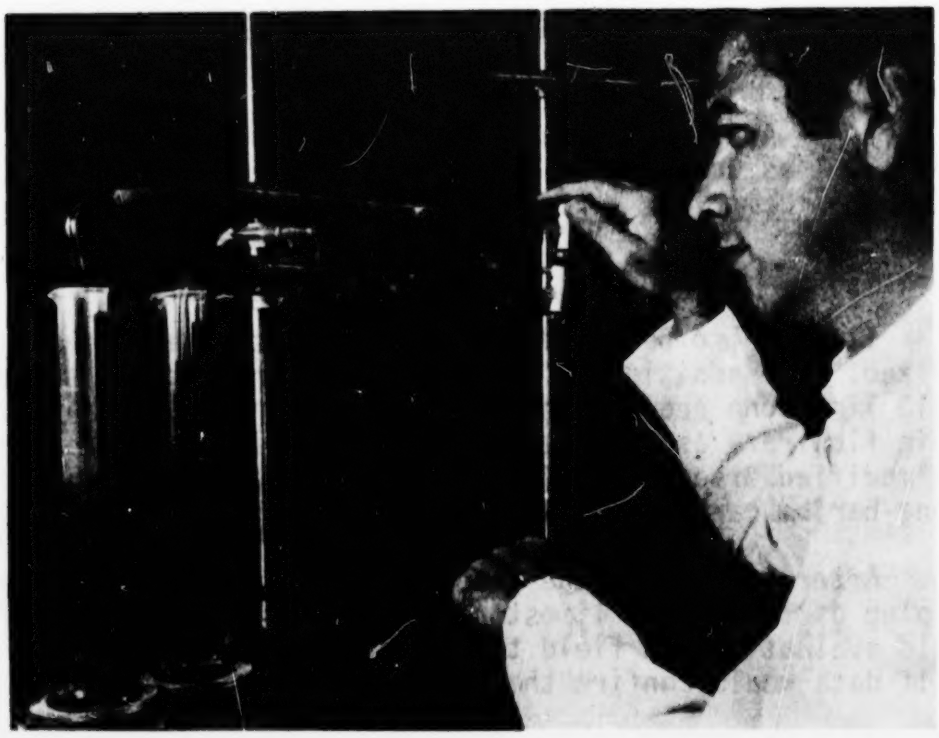


FIGURE 35

LABORATORY APPARATUS FOR PERMEABILITY STUDIES

Flow rate of acid mine water through the Harrold #12 graded limestone was set initially at 141 cc/hr to correspond to a retention time of 23 minutes. A constant flow rate was very difficult to maintain in the laboratory test due to precipitation of iron hydroxide as the water was neutralized; however, tests were able to confirm the retention time as sufficient to neutralize the acid mine water. The variation in pH is due to fluctuation in the flow rate. After 209 hours, it was not possible to continue any flow through the test column (Table 19 - Appendix).

Additional laboratory tests were conducted using the same test apparatus to determine the length of time it would require to plug a limestone column with less permeability. A constant head of acid water was maintained to exert a pressure of 2 psi on the limestone column. Water samples were analyzed at various times during the test. Flow rate was set initially to correspond to a retention time of 23 minutes. This flow rate gradually decreased during the test until no flow could be obtained through the limestone due to precipitation of iron hydroxide.

The modified Greer #13 was tested. It completely plugged in 125 hours. The same material with 1% barium carbonate added was also tested with similar results (Tables 20 and 21 - Appendix).

An analysis of the acid mine water used in these tests revealed the following characteristics:

pH	Acidity as $CaCO_3$ (mg/l)	Alkalinity as $CaCO_3$ (mg/l)	Total Iron (mg/l)	Dissolved Iron (mg/l)	Ca++ (mg/l)	SO_4 (mg/l)
3.2	180	0	30	25	500	1450

It can also be noted that in each test the acid mine water was neutralized. The addition of the 1% barium carbonate to the modified Greer #13 limestone reduced the permeability and caused an immediate decrease in flow rate as compared to a gradual decrease throughout the test for the modified Greer #13. This caused slightly higher pH values with the limestone-barium carbonate mixture, but did not increase the sulfate.

After evaluating the laboratory test results, it was decided that a plug of the graded limestone should be placed in a drift opening for field evaluation. A field test was made under controlled conditions to see if data would confirm the laboratory results.

Mine No. 40-085 was selected for the site of the installation. This was an open drift mine in Mount Clare, Harrison County, West Virginia which had not been affected by stripping. Water was flowing from an 8 x 5 foot opening in excess of 20 gpm into Browns Creek. The pH of the water was 3.3.

The ground was excavated to receive a wooden trough in the discharge flow path. The trough was constructed in the excavated area using wood framing and marine plywood. The finished trough was 5 feet high, 6 feet wide and 30 feet long. Three separate compartments 2 feet wide were provided across the width, with 3-inch standpipes in each compartment at the center and input end. Standpipes had the lower 30 inches perforated, enabling the height of water to be measured at each point. The standpipe next to the inlet reservoir was designated as number 1 and the pipe in the center as number 2. A prefix was added to designate the compartment number. Thus, location 1-2 would be the center standpipe of compartment number 1. The completed trough is shown in Figure 36 before filling with limestone.

The drain end of the trough contained a 3-inch drain hole at the lower part of each compartment. The trough was filled with Harrold #12 limestone. A flume 16 inches square was made to bring the water from the mine discharge into an input reservoir on one end of the trough. The inlet connections were installed with a valve on each compartment to regulate the input flow. In Compartment No. 1, a 3-foot extension pipe was placed below the valve so fluid discharged 2 feet above the bottom of the trough in the limestone bed. In Compartment No. 2 (center) the fluid was injected at the top of the compartment. In Compartment No. 3, a 2-inch plastic pipe 20 feet in length was placed on the top of the limestone and connected to the inlet valve. Holes 1/2-inch in diameter were drilled at intervals along the pipe in order to distribute 10 gpm of acid mine water along the length of the limestone bed. Figure 37 shows the inlet of the trough with the valves and inlet standpipes clearly visible. A 4-inch line was installed above the top of the bed level as an overflow line, and an 8-inch line installed to bypass surface water around the installation.

Flow was initiated through the trough. It was quickly evident that the water would not readily flow out the discharge end of the compartments, so the end of the trough was removed and the limestone allowed to take a natural angle of repose. The top was covered with plastic to prevent entry of water other than mine discharge.

The flow was then monitored through each compartment and at each standpipe. The input flow was set at 9.4 gpm for each compartment (Table 22 - Appendix). Figure 38 is a graph of the flow through each compartment. It can be noted that the flow decreased steadily in each compartment. Compartment No. 2 overflowed into adjoining compartments on the twelfth day, and Compartment No. 1 overflowed the next day. At this time, flow input on No. 1 was changed to top input similar to that in No. 2 and input flow readjusted into these two compartments to provide maximum flow possible without overflow.

Throughout the balance of this test, it was necessary to clean and adjust the valves periodically to maintain flow through the trough.

The water level in the limestone as measured by the standpipes is also shown in Table 22. A comparison of the level in each compartment at several times during the test is graphically illustrated in Figure 39. The numbers of the lines show the sequence of measurements taken throughout the test.

It will be noted that in Compartments No. 1 and No. 2, the flow pattern was very similar. The height of flow increased at standpipe 1 and decreased at standpipe 2. In Compartment No. 3, the flow was more uniform along the length of the trough than in the other two compartments as indicated by the height decreasing in both standpipes. More flow was required to effect a plugging in the limestone. Whereas Compartments No. 1 and 2 plugged with precipitate in 12 and 13 days as indicated by overflowing, Compartment No. 3 did not completely stop flowing during the test. The flow through Compartments No. 1 and No. 2 after plugging occurred was undoubtedly due to leakage from Compartment No. 3.

The test was stopped after 49 days. Evaluation of the test was difficult since considerable leakage was noted from the wooden trough and possibly from one compartment to another. In addition, plugging was continually encountered in the three inlet valves as well as the discharge holes in the plastic pipe used along the length of Compartment No. 3. Since it was necessary to clean and readjust the valves several times, it was not possible to maintain a constant input rate. However, the graded limestone plug did gradually seal due to precipitation of iron hydroxide which plugged the pore space in the limestone.

In the compartments where the water was introduced into the front end of the #12 limestone, the plugging occurred for the full height in the first few feet adjacent to the water input point, and gradually declined to approximately 6 inches height just before discharge from the limestone. In the compartment where the water was introduced along its length through a perforated pipe, the plugging occurred for the full height throughout the 20-foot section covered by the pipe. This can be noted on Figure 40. This drawing depicts the plugged area of each compartment as determined by the stain from the iron floc precipitated during the neutralization process.

It was also found that the water passing through the plug was partially neutralized and iron content reduced. The average acidity for the mine water in this test was 568.5 mg/l. The average reduction in acidity by passage through each compartment ranged from 81% for Compartment No. 1 to 89% for Compartment No. 3. The average iron content for the mine water was 125.9 mg/l. The average reduction in iron during the test varied from 48.5% for Compartment No. 1 to 53% for Compartment No. 2 and 3. The mine water pH of 3.2 to 3.8 was changed to a pH of 5.7 to 7.0 by passage through the 30 feet of limestone. Analyses of all water samples obtained in this test are included as Table 23 in the Appendix.

A sample of Harrolds #12 limestone which was coated with sludge was taken from the trough for analysis to determine the volume and com-

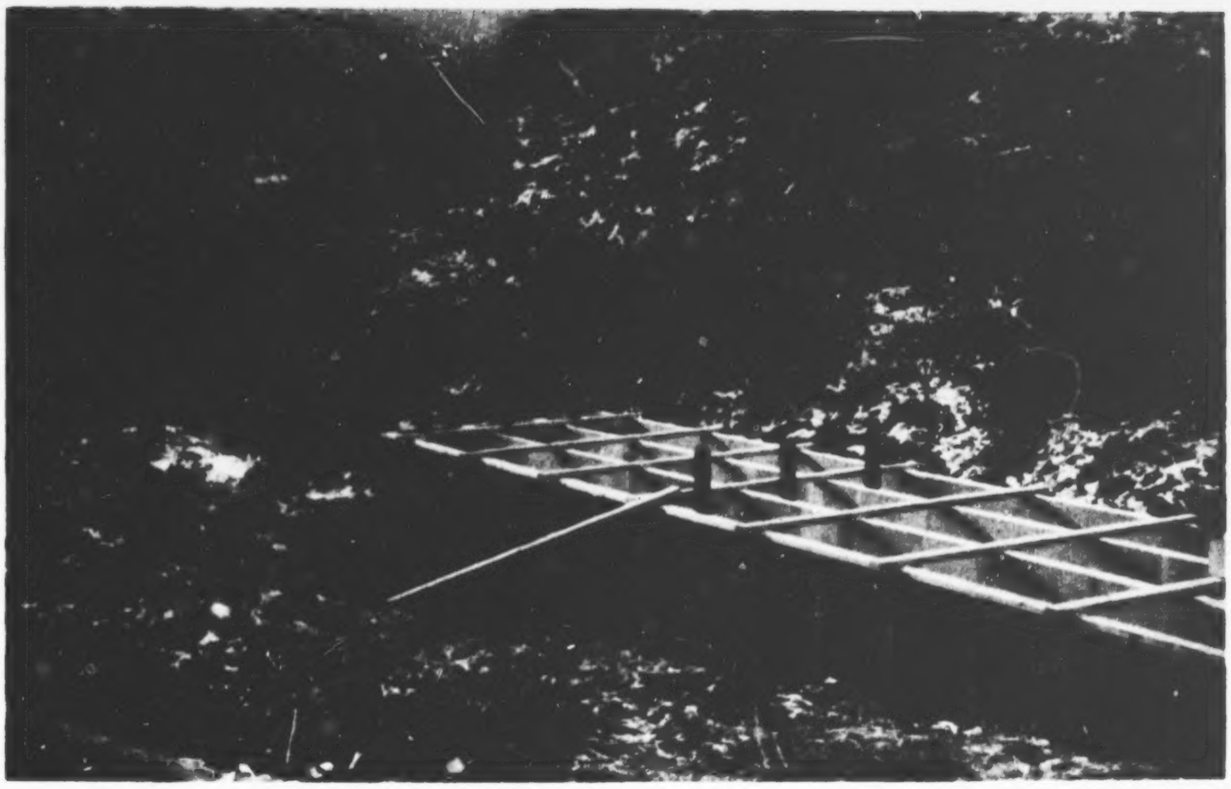


FIGURE 36 - COMPLETED TROUGH - PERMEABLE PLUG TESTS

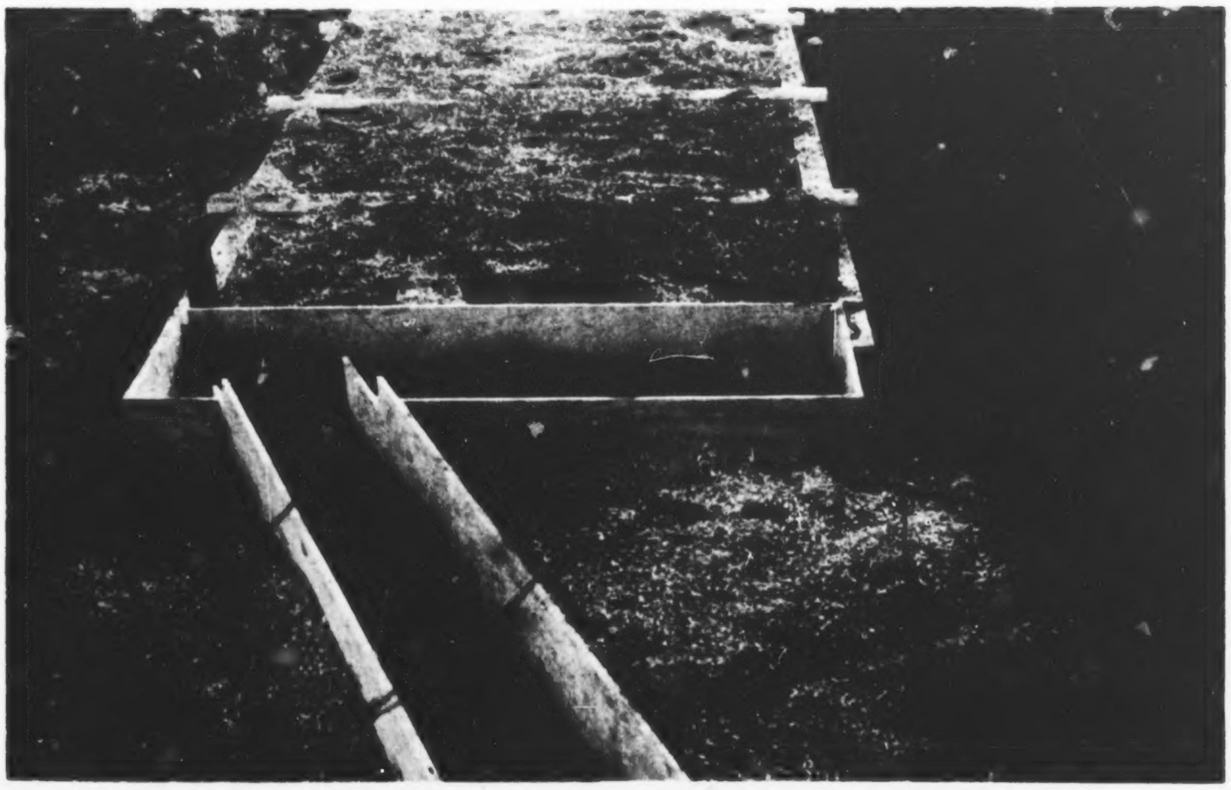
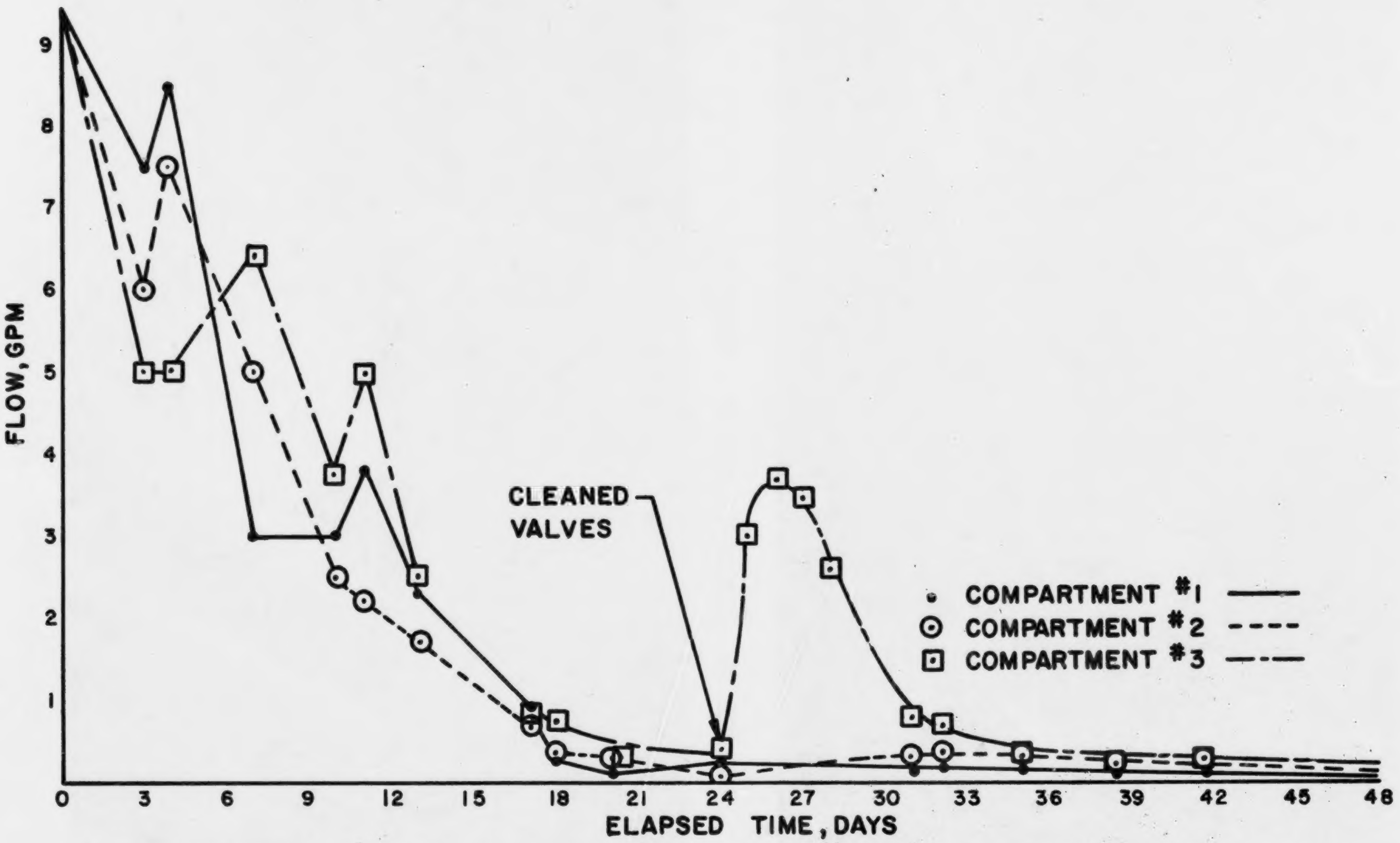
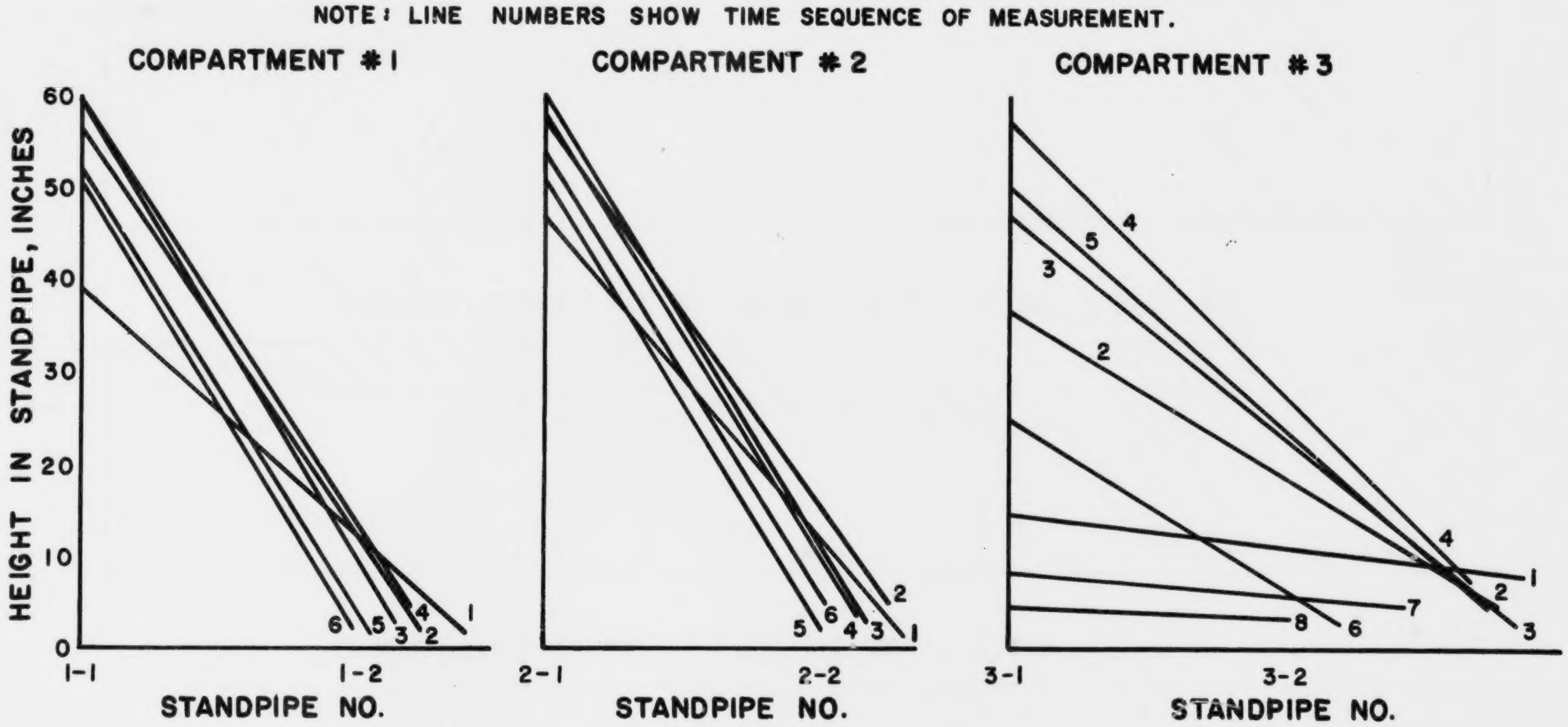


FIGURE 37 - TROUGH - INLET COMPARTMENT AND VALVES

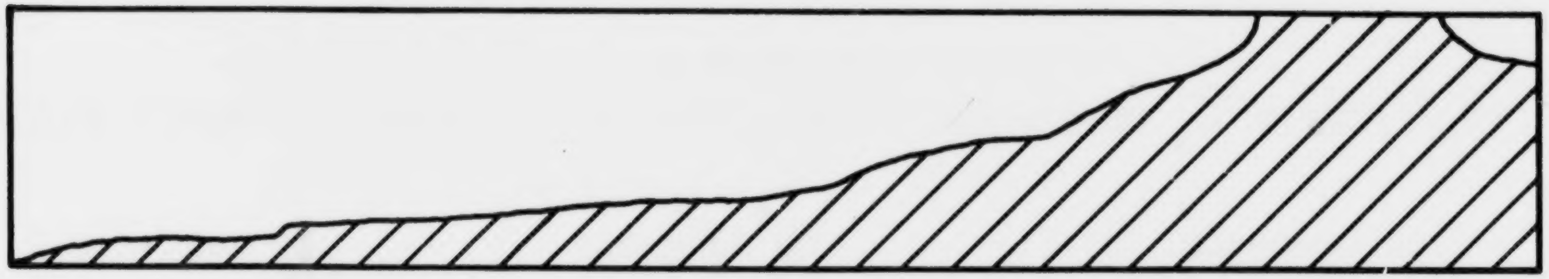


FLOW THROUGH LIMESTONE IN WOODEN TROUGH - PERMEABLE PLUG TEST NO. 1
FIGURE 38

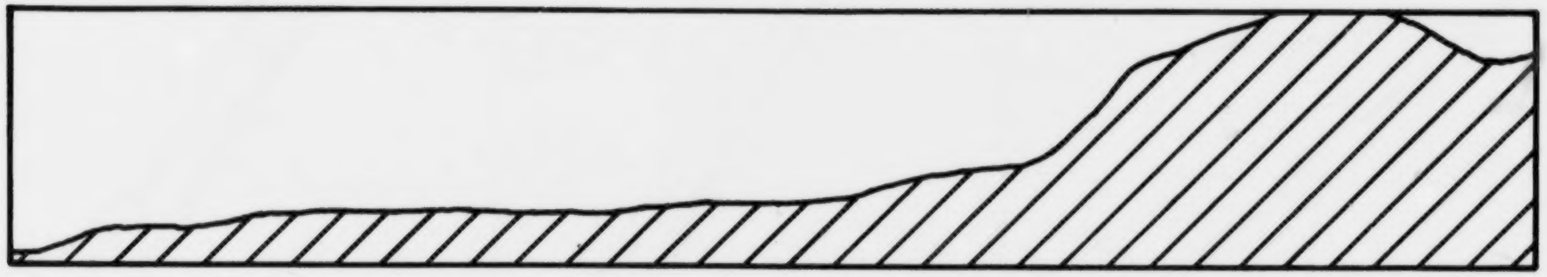

NOTE: LINE NUMBERS SHOW TIME SEQUENCE OF MEASUREMENT.

WATER LEVEL IN LIMESTONE FILLED TROUGH - PERMEABLE PLUG TEST NO. 1

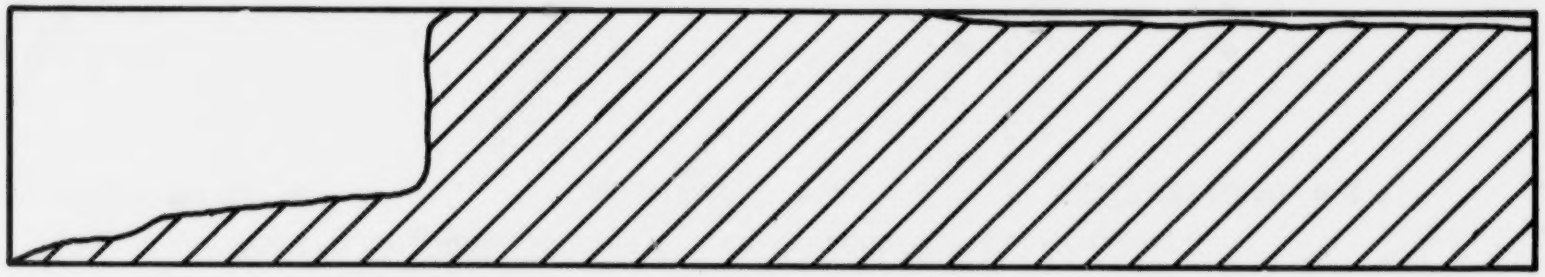
FIGURE 39



COMPARTMENT NO. 1 - PERMEABLE PLUG TEST NO. 1



COMPARTMENT NO. 2 - PERMEABLE PLUG TEST NO. 1



COMPARTMENT NO. 3 - PERMEABLE PLUG TEST NO. 1

FIGURE 40 - PLUGGING PROFILE - WOODEN TROUGH - TEST NO. 1

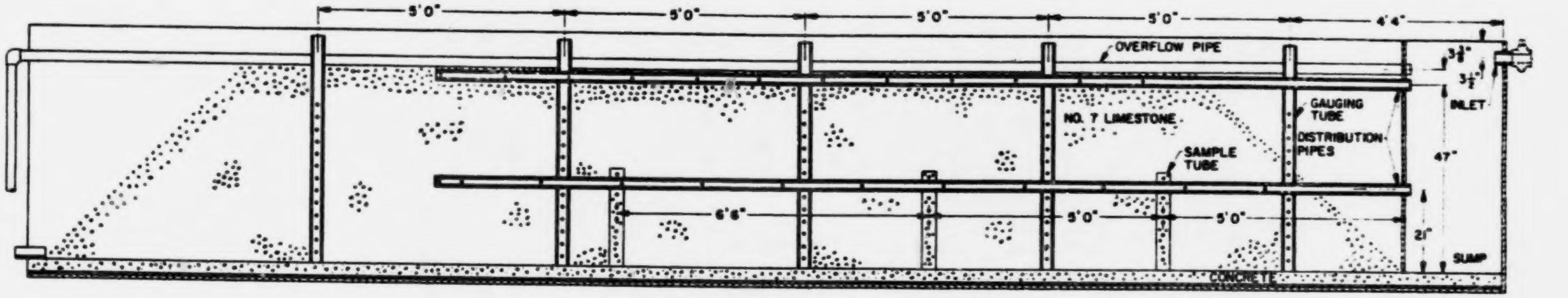
position of the sludge formed during the test. It was determined in laboratory examination that 12 cc of sludge were deposited for each 100 cc of limestone in the particular sample submitted, but this could vary somewhat depending on where and how carefully the sample was taken. Based on this analysis, approximately 30 cubic feet of sludge would be obtained for each compartment in 49 days. The X-ray analysis showed the sludge coating to be primarily amorphous iron oxide with a small amount of quartz and iron carbonate.

After the evaluation of these test results, it was decided to conduct an additional test in the wooden trough. The purpose was to confirm the previous test results with #12 limestone and to evaluate a limestone in the same manner which had a higher permeability.

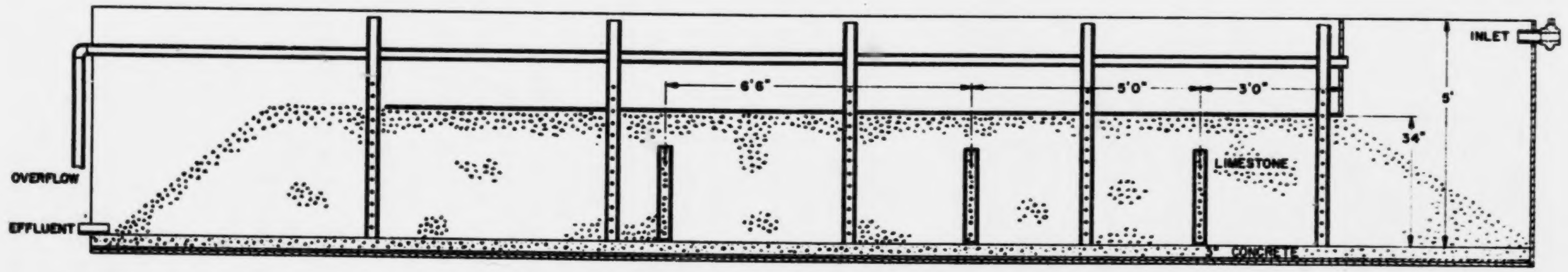
In order to eliminate the leakage problem in the wooden trough, the compartments were cleaned out and then sealed. Sealing was accomplished by grouting beneath the trough with cement slurry to fill all voids, caulking all seams in the compartment with a caulking compound and placing a 3-inch concrete floor in the trough. In addition, the compartments were lined with a layer of 6 mil plastic sheeting before filling with limestone.

Each compartment contained three sample tubes located strategically along the length of the trough. These tubes were 3-1/2-inch OD plastic 24 inches long, perforated with 1/2-inch holes at 6-inch centers on two sides. Tubes were filled with limestone identical to that in the compartment and ends capped. They were then placed in the compartment in an upright position as the compartment was filled with limestone. Gauging pipes were made of 2-3/8-inch OD plastic with 7/16-inch holes drilled at 3-inch centers on two sides over the lower 30 inches. These were placed at five points in each compartment with 5 feet between pipes. A drawing showing a section of each type of compartment is attached as Figure 41. Compartments No. 1 and No. 3 each had a roof over the limestone at a height of 36 inches above the floor. Compartment No. 1 contained Harrold #12 limestone and Compartment No. 3 held Harrold #7 limestone. Compartment No. 2 contained Harrold #7 limestone to a depth of about 4 feet. Two 2-3/8-inch OD plastic distribution pipes, 20 feet long with 1/4-inch holes on 10-inch centers, were placed in the limestone. One was 21 inches above the floor and the other was 47 inches.

Figure 42 shows a view of the wooden trough as used in this test. The roof and protruding gauging pipes can clearly be seen in the two outer compartments, as well as the limestone and gauging pipes in the center compartment. Figure 43 shows the discharge end of the trough. The large by-pass pipe is shown on the left side. The input arrangement for the trough is shown in Figure 44. Note that the flow comes from the input reservoir into a sump in each compartment, and then flows through the limestone or into the distribution pipes. The amount of flow desired can be obtained by adjustment of valves in the input reservoir. Any excess flow above that desired will go through the large bypass pipe on the outside of the trough.


5'0"
5'0"
5'0"
5'0"
4'4"
OVERFLOW PIPE
3 3/8"
3 1/2"
INLET
GAUGING TUBE
DISTRIBUTION PIPES
NO. 7 LIMESTONE
SAMPLE TUBE
6'6"
5'0"
5'0"
47"
21"
SUMP
CONCRETE

COMPARTMENT NO. 2 - TEST NO. 2


INLET
6'6"
5'0"
3'0"
5'
34"
LIMESTONE
OVERFLOW
EFFLUENT
3" CONCRETE

COMPARTMENTS NO. 1 & 3 - TEST NO. 2

SECTION - COMPARTMENTS - WOODEN TEST TROUGH

FIGURE 41

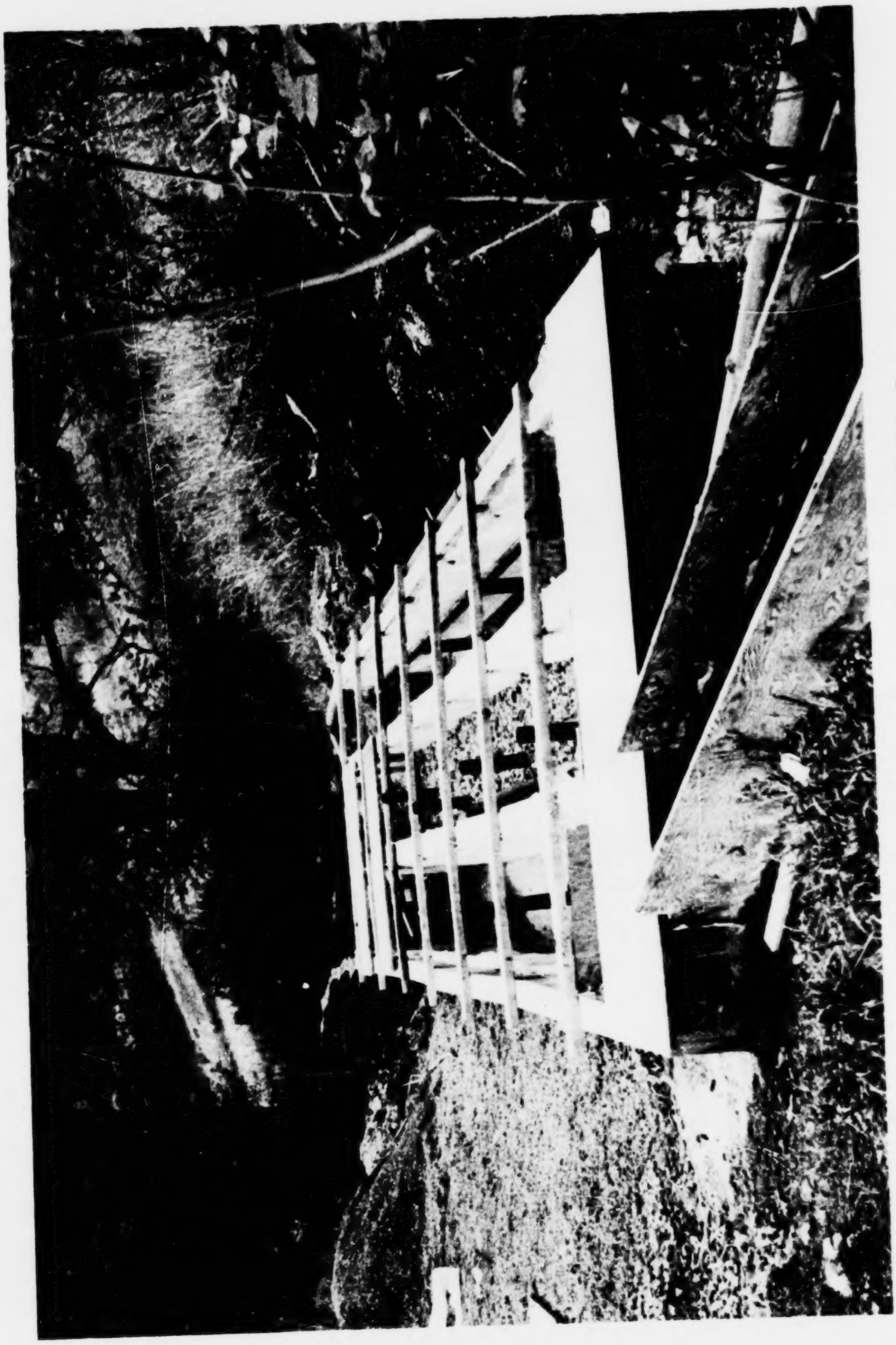


FIGURE 42 - TROUGH - OVERALL VIEW - TEST NO. 2

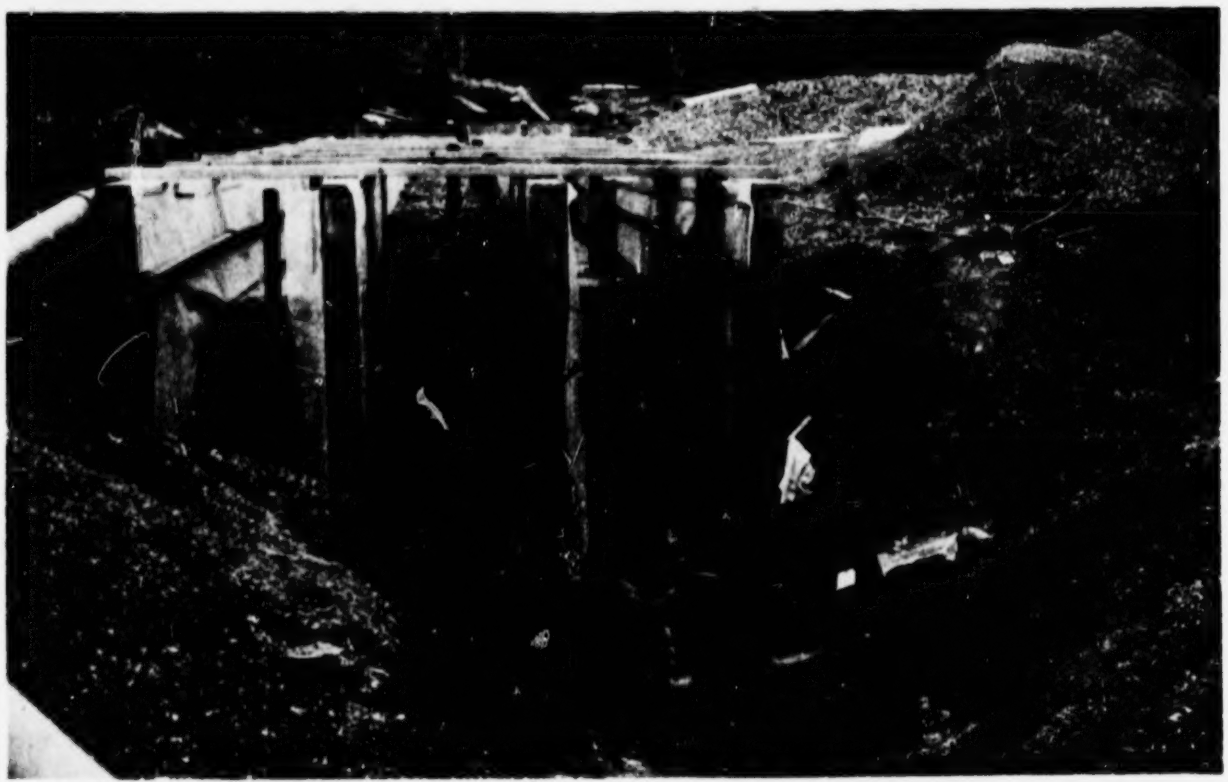


FIGURE 43 - TROUGH - DISCHARGE END - TEST NO. 2

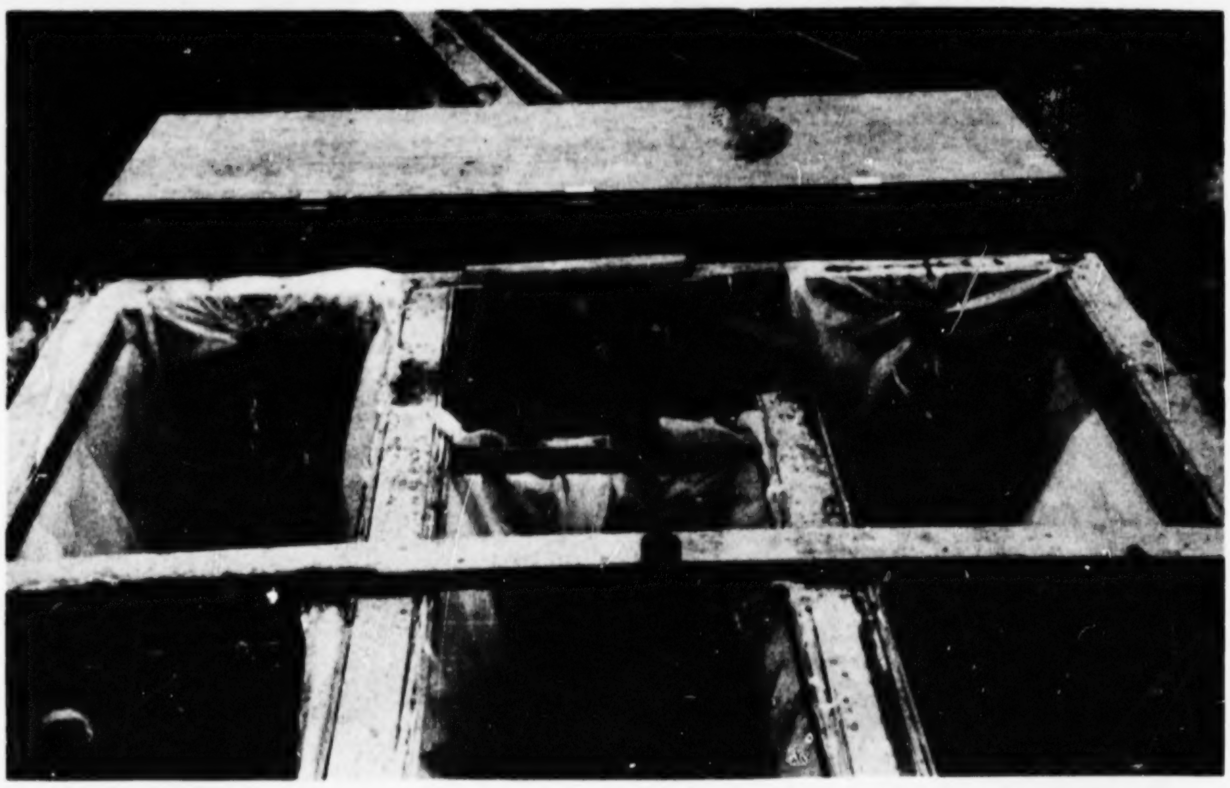


FIGURE 44 - TROUGH - INPUT ARRANGEMENT - TEST NO. 2

The flow test was initiated and monitored to obtain flow rate and water samples. Fluid levels were monitored from the third day of the test, (see Table 24 - Appendix). It was found that Compartment No. 1, containing Harrold #12 limestone, plugged and overflowed as in Test No. 1. While maintaining a 7 gpm input rate, the discharge rate declined steadily throughout the initial 30 days of monitoring to 0.5 gpm.

Compartment No. 3, identical in construction to No. 1, contained Harrold #7 limestone. After 5 weeks, the discharge rate was still equaling the input rate in this compartment, although the water level in the gauging pipes had risen considerably at every point. However, the discharge rate began to decline the following week. Compartment No. 2, also containing #7 limestone but having the water introduced through distribution pipes, followed a similar decline in discharge to Compartment No. 3.

An extension was added to the sump of Compartment No. 1 to provide for an increased hydrostatic head on this roofed portion containing #12 limestone. The sump was 8 feet high. A 2-foot head was placed on the compartment and flow started again. This head was maintained and a discharge of 3 gpm recorded on the following day. The discharge continued to decrease. Raising the head to 3 feet did not increase the discharge. It is believed that the increased initial discharge was due primarily to breaking the seal in the aggregate pores by the construction of the sump extension.

After 3-1/2 months of monitoring the flow, the test in Compartment No. 2 was discontinued since the flow rate was then down to 0.4 gpm with 7 gpm input. A sump extension was built on Compartment No. 3 similar to that already in use on Compartment No. 1. Tests with heads up to 4 feet did not show any increased flow through the limestone. (Table 25 - Appendix).

The water analyses (Table 26 - Appendix) indicated that the water passing through the #12 limestone with an initial pH of 3.1 neutralized to a pH of about 5.8. The water flowing through the #7 limestone remains essentially unchanged, probably due to less retention time.

It was then determined to conduct permeable plug Test No. 3. This test was the placement of #12 limestone in an actual mine to form a self-sealing plug. Prior to actual field placement, a full-size laboratory test was conducted to build a plug in a simulated mine passage. This mine passage was of plywood construction with a section 10 feet wide by 6 feet tall, and a length of 16 feet.

The #12 limestone was conveyed pneumatically from a 220-cubic-foot pressure tank through 60 feet of 4-inch hose and 20 feet of 4-inch steel pipe. A 45-degree deflector plate at the discharge end of the pipe was used to guide proper placement of the limestone, as can be seen in Figure 45. This feature permitted the operator to control the building of the plug. Material was conveyed into the tunnel at the rate of about 835 pounds per minute, but placement, including loading time, reduced the overall rate to about 550 pounds per minute. The finished plug was about 20 feet at the base with a one-foot closure at the top side. It was not possible on this test to obtain a tight connection between the roof and aggregate. The plug had an angle of repose of approximately 45 degrees on the front side and 30 degrees on the back side. About 33 tons of rock were used in building the plug.

Arrangements were then made to build a plug in a mine passage. A drift opening in Mine No. 62-008 near Clarksburg, West Virginia, was selected for the test site. This opening was the left-hand Pittsburgh portal, designated as Opening No. 3. It may be seen in Figure 46 and is also denoted on the schematic drawing shown as Figure 34. A roof fall was evident about 100 feet back into the mine drift. A buildup in the floor about 32 inches high was noted across the drift in the vicinity of the roof fall. These are evident in Figure 47. This picture also shows the numerous posts existing in the mine area where the bulkhead was built.

A trailer having two 250-cubic-foot pressure tanks was used to contain the #12 limestone. The rock was loaded into the tanks from a dump truck using a belt conveyor system. Both tanks were loaded prior to the job, then as one tank was emptied, it was refilled while the material was being placed from the other tank. This allowed semi-continuous operation, stopping only briefly to move connections from one tank to the other.

The #12 limestone was brought from the quarry in dump trucks with the tail gate altered to provide an 18-inch square opening and chute to discharge on the belt conveyor. The total weight of limestone used was 134,200 pounds.

The first half of the stone delivered was freshly crushed and dry. This material moved well pneumatically through the 80 feet of 4-inch hose and 30 feet of 4-inch aluminum pipe used in placement. However, limestone dust in the mine portal was very bad and hindered proper observation of the work.

The second half of the aggregate was obtained from stockpiles due to a breakdown of the crusher. This limestone was saturated with water because of recent rains and did not cause any dust problems, but it was difficult to move pneumatically because of build-up in lines and valves of the fine limestone particles in the aggregate.

Material was moved from the tanks using a maximum pressure of 35 psi, assisted at times by an air vibrator attached to each tank. A special air manifold on the tank, discussed in a previous section, was used to facilitate movement of the aggregate from the tanks.

Figure 48 shows the seal with about one-half the limestone in place. The placement pipe with deflector on the end, as well as the drain line, can be seen. The angle of repose is about the same as that of the simulated mine test, or about 45 degrees.

When stoppages were not encountered, a tank of about 17,000 pounds could be emptied in an average time of 20 minutes. This is a rate of 850 pounds per minute.

The finished seal of aggregate filled the 52-inch by 12-foot drift with 25 feet of roof contact and 36 feet of length on the base. It was observed that some settling occurred after several days.

Monitoring of the water flowing through the limestone plug was begun immediately. The flow data and water analyses for this mine drift are attached (Table 27 - Appendix). It can be noted that the water was neutralized from a pH of 3.0 to a pH of 6.3 to 6.9 with a corresponding change in acidity from 300 mg/l to an average of 112 mg/l. The drainage of 3 gpm from the drift did not show any major decline during the four months of monitoring. By comparison, a flow of 7 gpm through the same aggregate in the test trough decreased to zero in 55 days. However, the aggregate volume in the test trough was only 12% of that in the drift, while the flow rate was more than double that through the larger plug. This should indicate that a much longer time would be required to form enough precipitate to close the pores in the larger plug installation.

An open fanway about 100 feet north of Opening No. 3 is draining and is interconnected to the three sealed openings. This fact may limit the head behind the seals to the approximately 50 inches shown in the table.

The cost for placing the graded aggregate seal in the mine drift was $3,048.00. This cost included $756.00 for site preparation, $237.00 for materials and $2,055.00 for equipment and operators.

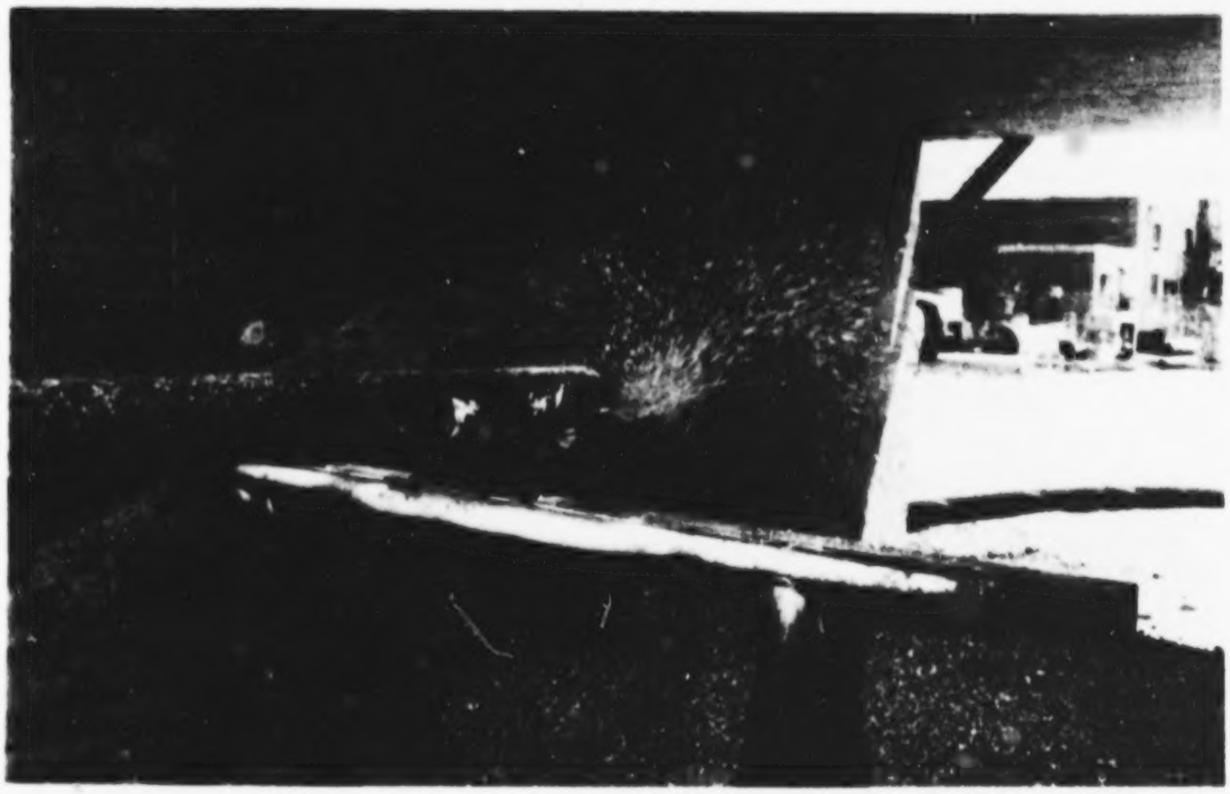


FIGURE 45 - PLACING AGGREGATE IN SIMULATED MINE OPENINGS

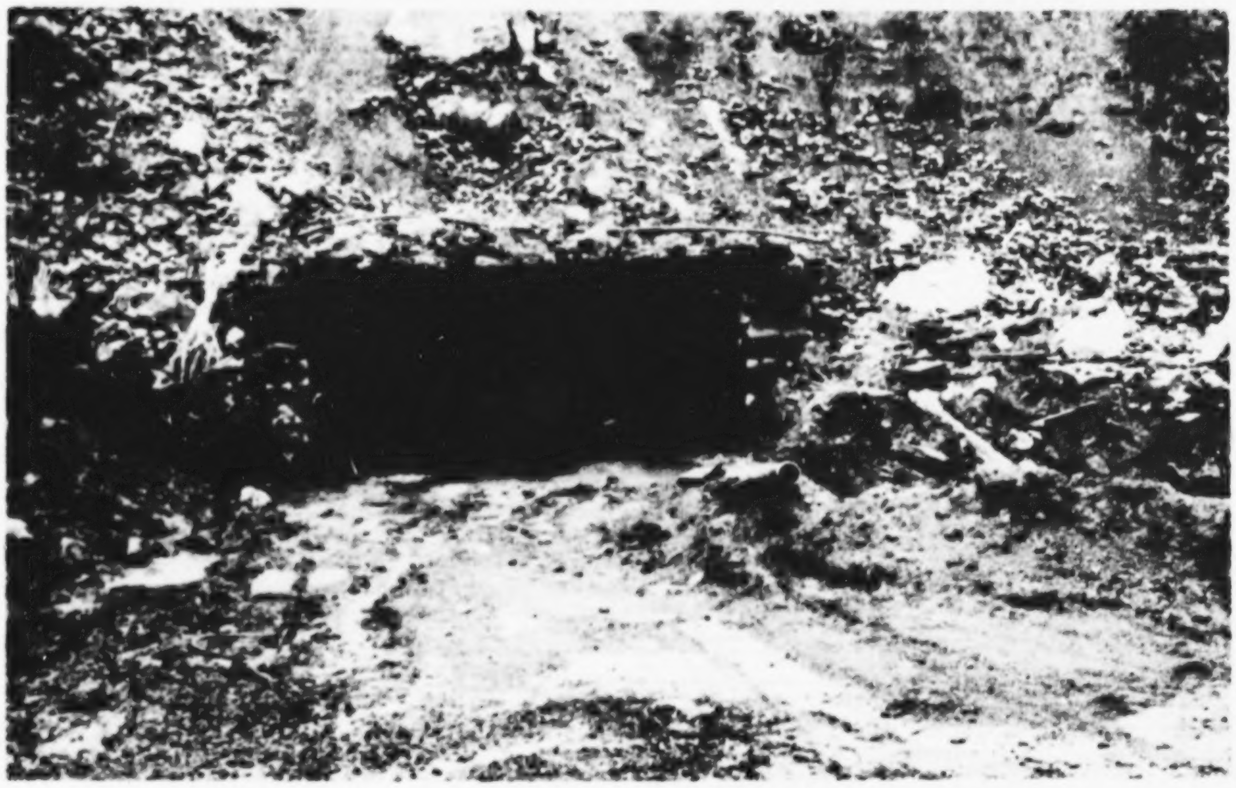


FIGURE 46 - DRIFT MINE FOR PERMEABLE PLUG INSTALLATION

FIGURE 47 - MINE INTERIOR - PERMEABLE PLUG SITE

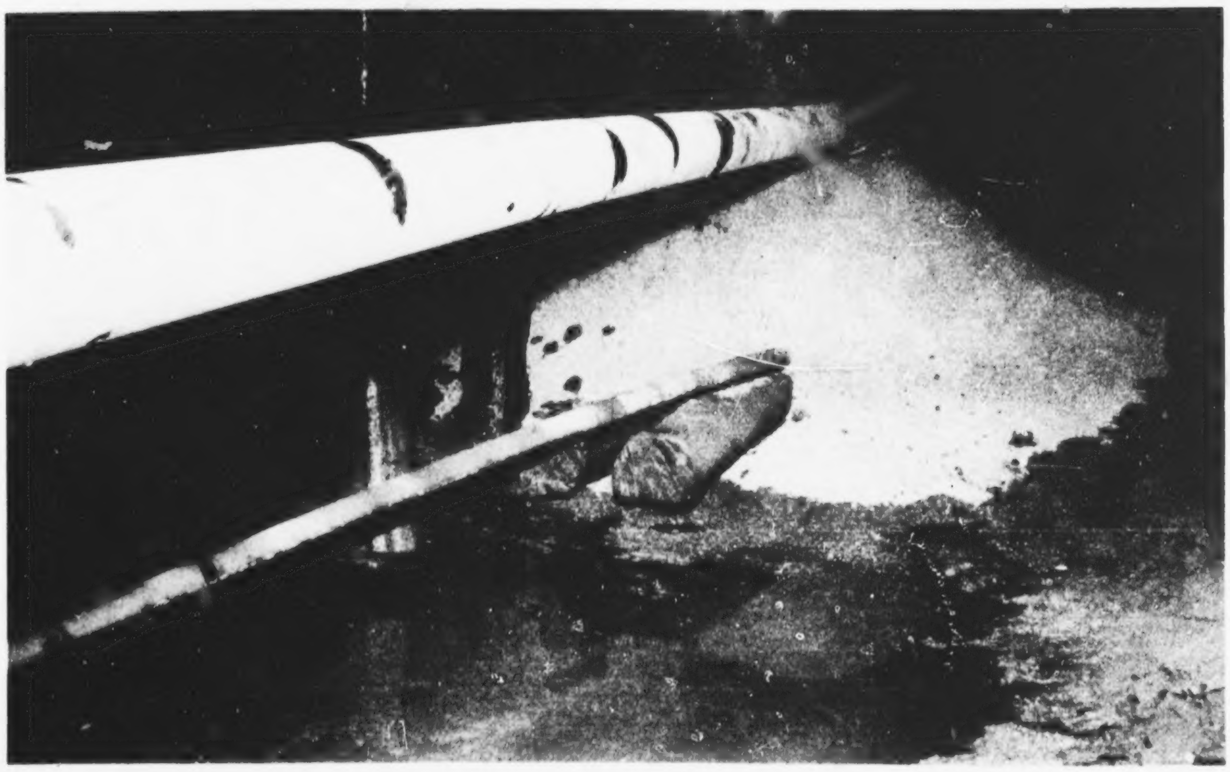


FIGURE 48 - PLACING LIMESTONE SEAL IN MINE DRIFT

SEALING OF HIGH-FLOW MINE

The purpose of this phase of the project was to develop methods whereby remote grouting application could be used in mine drifts having high water discharge rates. The techniques and materials which were developed through research in another phase of this project were applied to the conditions found in the high-flow mine.

Mine No. RT5-2, near Coalton, West Virginia in the Elkins project area, was selected for the field tests with the cooperation of the personnel at the Norton FWPCA laboratory. The location of this mine is shown on the map attached as Figure 49. A mine map of the Coalton No. 3 Mine in the Coalton-Norton Coal Field is attached as Figure 50. The opening selected for field tests is designated on the mine map as Opening A.

Drift mining began in this field in 1894 when the Coalton No. 1 Mine was started by an individual using pick and shovel mining methods. Four years later it was taken over by the Davis Coal and Coke Company, who constructed a large power plant for the mine and 500 coke ovens during 1900. Hand mining continued using short wall machines and, later on, arc wall machines. All haulage to the main haulways was accomplished by horses. The first recorded production was 220,040 tons of coal and 87,907 tons of coke in 1908. The maximum production of this mine was 269,017 tons of coal in 1915 just prior to ownership changing to the West Virginia Coal and Coke Company. Coal production declined to 2,769 tons in 1937.

The Coalton No. 2 Mine was started in 1900 and abandoned in 1928. The Coalton No. 3 Mine, in which the field tests occurred, began in 1904 to 1905 and operations ceased in 1923. All of these Coalton Mines were hand mined and pillared during retreat mining. The remainder of the coal in Coalton No. 3 Mine was mined up to 1950 and was hauled out of the Norton No. 2 Mine.

Norton No. 1 Mine was started in 1902 by Davis Coal and Coke Company who also owned the Coalton Mines. The first recorded coal production at Norton was 255,792 tons in 1918. Steel pan conveyors were installed in 1920 in the Norton No. 2 Mine. This was supposedly the first mechanized mine in the United States. The maximum production of the Norton Mine was 540,403 tons in 1942.

There are presently two active mining companies in the area. The Demott Coal Company, which started in 1952, and Roaring Run Coal Company are operating in the Norton No. 2 Mine. Roaring Run Coal Company produces approximately 400 to 500 tons per day. During the 1940's the entire area was strip mined.

The seam mined in all these mines is in the Allegheny series and is called the Kittanning Seam. A geologic section of Mine No. RT5-2 is included in the Appendix as Figure 66 to show the formations which occur in the area. This mine is in the Elkins project area where

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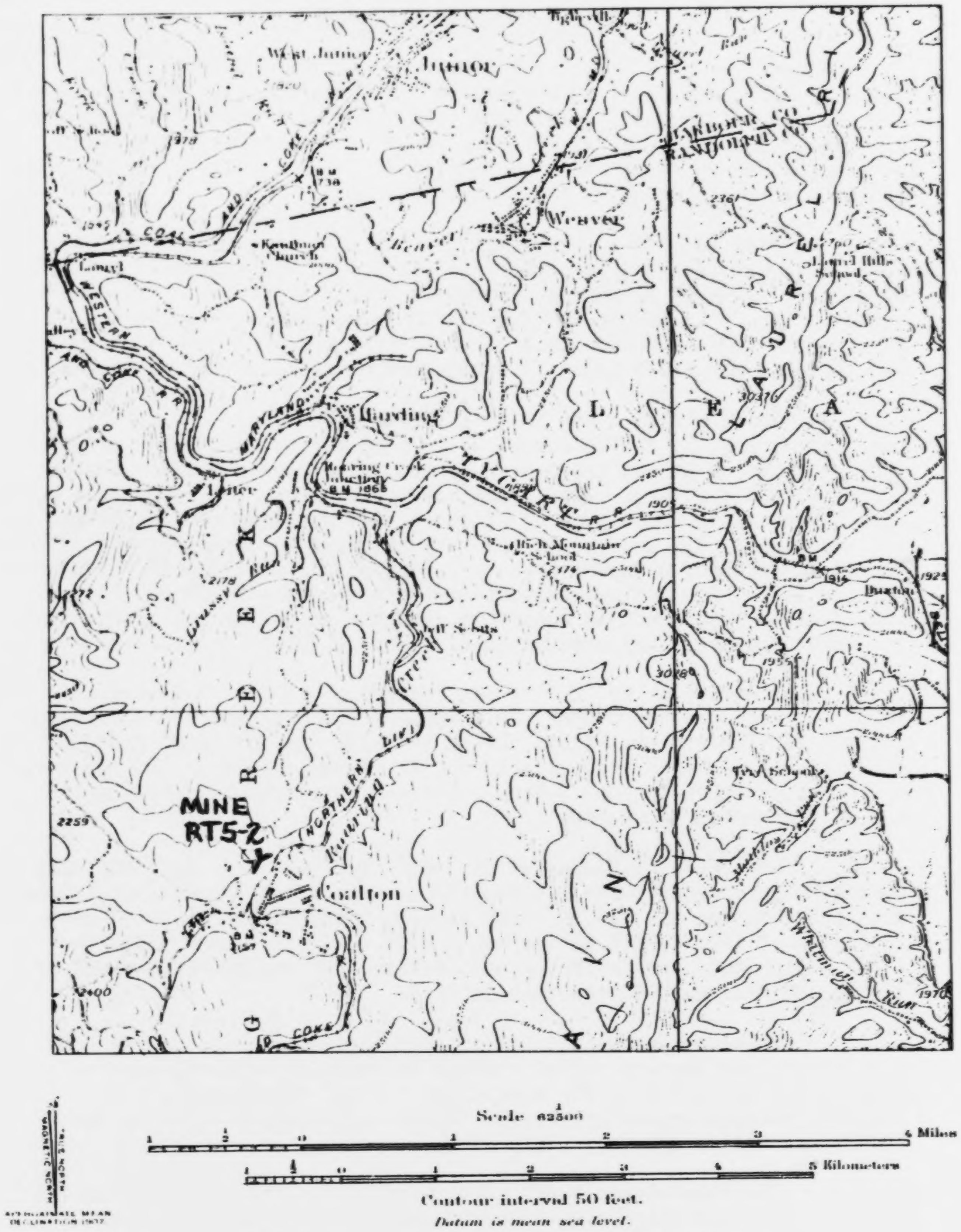

MINE
RT5-2
Junior
West Junior
Weaver
Kaufman Church
Harding
Roaring Creek Junction
B.M. 1868
Rich Mountain School
Laurel Hills School
Buxton
Coalton
Beaver
Grassy Run
Roaring Creek
Whitman Run
WESTERN
MARYLAND
COAL AND COKE R.R.
NORTHERN DIV.
BARBOUR CO.
RANDOLPH CO.
L E A
L A U R E L
G R E E K
N
2178
2259
2400
2361
1905
1970
1914
1925
3037
Scale $\frac{1}{62500}$
1
½
0
1
2
3
4 Miles
5 Kilometers
Contour interval 50 feet.
Datum is mean sea level.
MAGNETIC NORTH
TRUE NORTH
APPROXIMATE MEAN DECLINATION 1907

FIGURE 49 - LOCATION MAP - MINE NO. RT5-2

considerable reclamation work was accomplished in 1966 and 1967 by the Federal Water Pollution Control Administration.

The field testing was accomplished in the following manner: The entire flow was diverted through a drain line placed on the floor of the drift. A rear bulkhead of quick-setting, self-supporting cementitious material was constructed in front of the wet seal. The drift was filled with aggregate after pipes were placed for a subsequent grouting operation. A front bulkhead of the quick-setting material was then placed across the front of the aggregate. The aggregate was grouted through the pipes placed for that purpose.

When the head of water impounded behind the seal increased, the pressure caused water to leak out from an unknown adjacent opening. Remedial work on the second opening was accomplished by constructing a bulkhead of graded aggregate and agricultural lime in the drift to seal the opening.

The two openings which were sealed were at one end of the Coalton No. 3 Mine, far from the main heading. These openings were originally driven out for use as air supply openings. Later these were used as entries. Since they were close to the stables in the town of Coalton, the horses were moved into the mine through one of these openings, and the other was used for a fanway. Opening No. 1 of Mine No. RT5-2, which was selected for the field test, was considered suitable because the site was easily accessible and provided sufficient room for the necessary equipment to be used in placing the mine seal. It also had a sound roof and good timbering and a water flow in excess of 50 gallons per minute. The coal beds have a slight dip and flooding of the entire section of the mine was thought to be possible. In addition to these qualifications, the natural drainage patterns surrounding this mine would tend to prevent the flooding of any populated area in case of leakage or failure of any opening in the mine. Other openings were known to exist and many of these were sealed only with a clay seal. A view of this opening prior to the field test is shown in Figure 51.

During the Elkins project, Opening No. 1 had been closed by placing a wet seal in the main shaft and a dry seal in the crosscut coming into the main passage from the right side. These are shown in Figure 52, which is a drawing of the remedial construction accomplished during the field tests. The wet seal consisted of a concrete footing and a double course of concrete blocks completely filling the mine drift. Openings had been provided through this wall by omitting two 8 by 16-inch blocks on each side of the mine passage, 16 inches above the floor of the mine. About 4 feet in front of this wall was a dam of concrete blocks which retained about 3 feet of water. This prevented air entry into the mine and permitted measurement of the water flow. The dry seal was a solid concrete block wall placed in a crosscut to the right of the passage about 4 feet back into the crosscut. To prepare this opening for placement of the mine seal, the dam in front of the wet seal was removed and a portion of the wet seal, including the

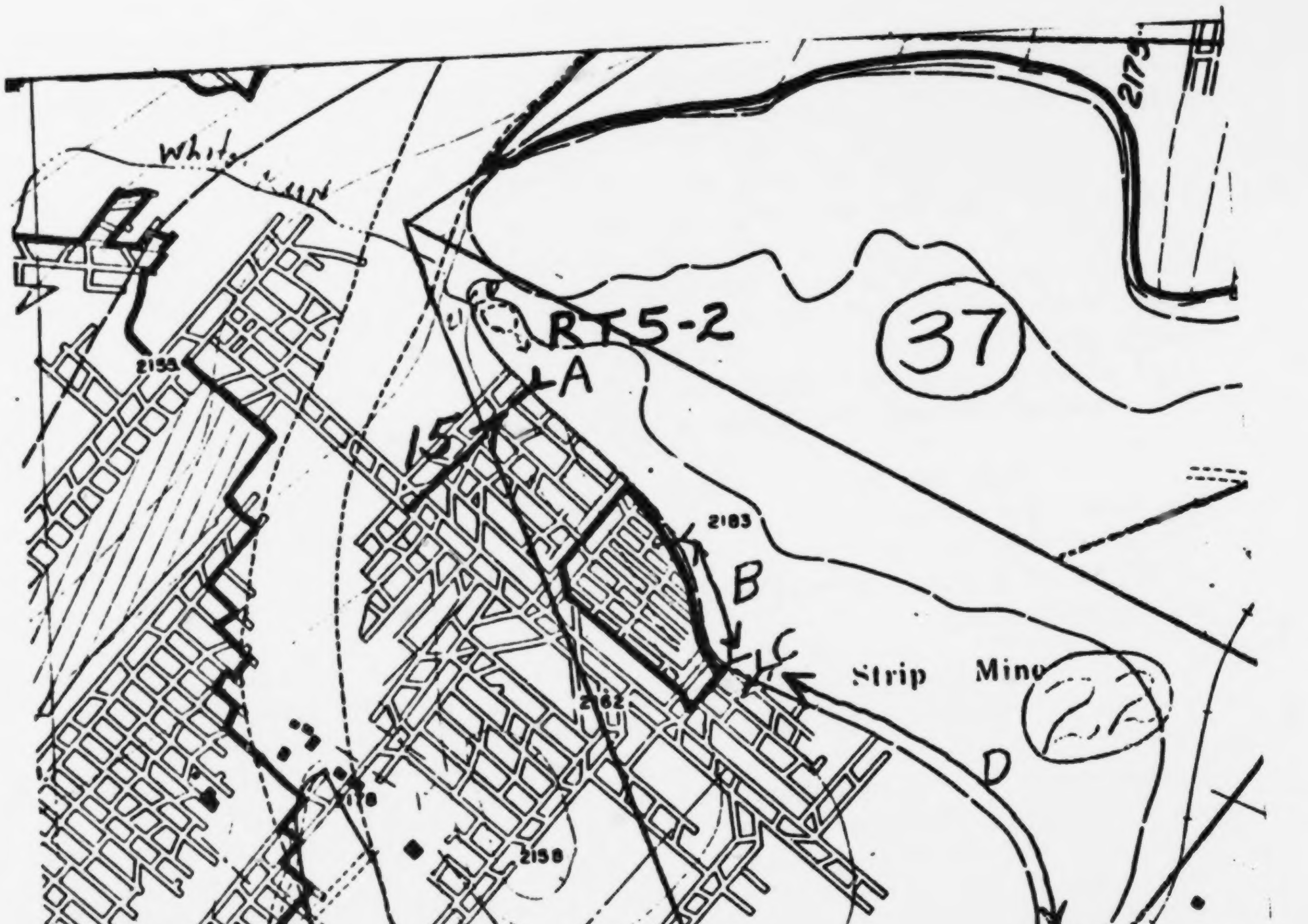

RT5-2
37
A
15
2183
B
C
Strip Mine
2162
D
2158
2155
E
F
2180
G
2176
Mining Unknown
H
2186
2188
2192
2196
M-19
2246.26
M-20
2230.03
21
COALTON
2175

FIGURE 50 - MINE MAP - COALTON NO. 3 MINE (RT5-2)

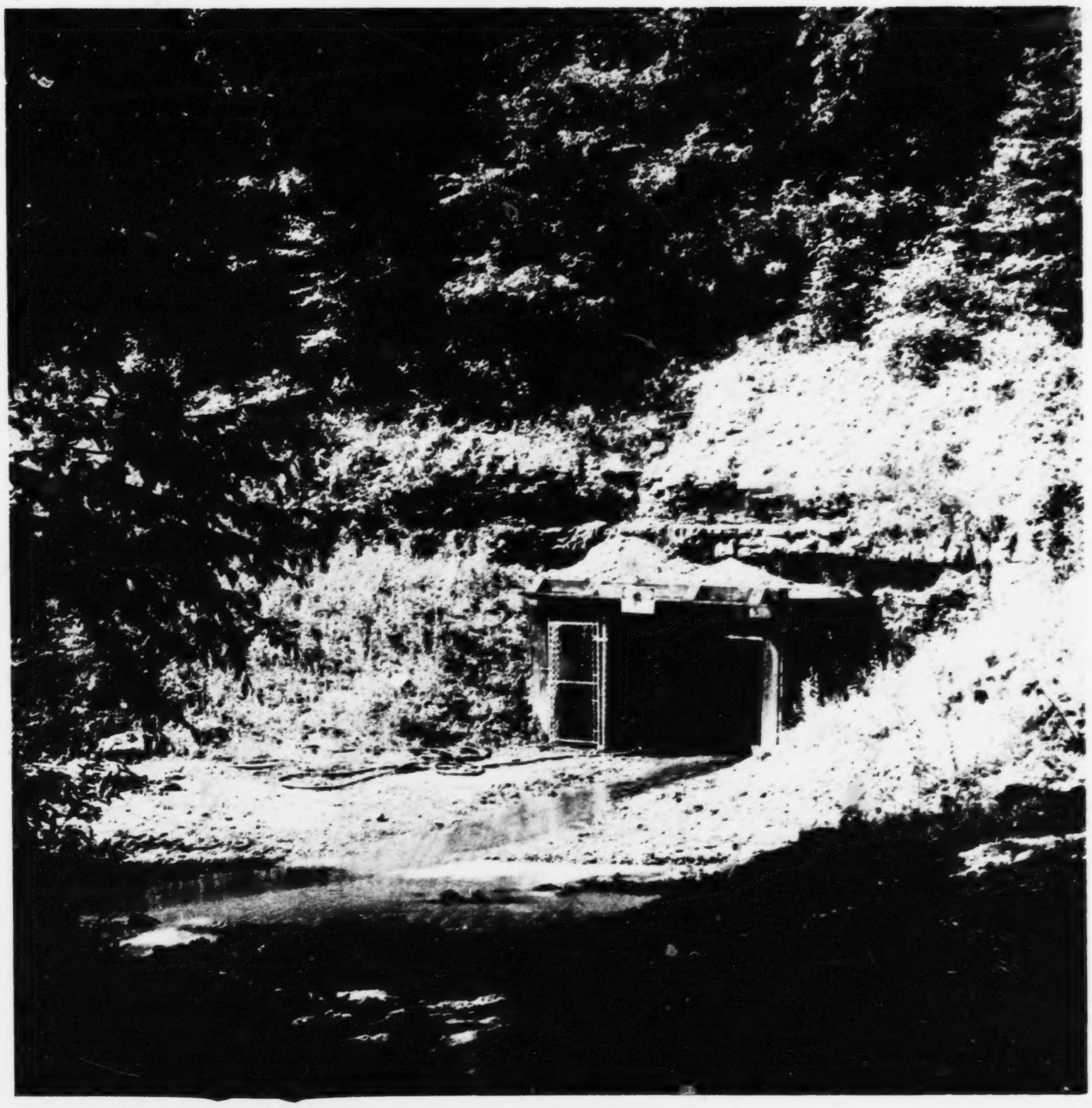
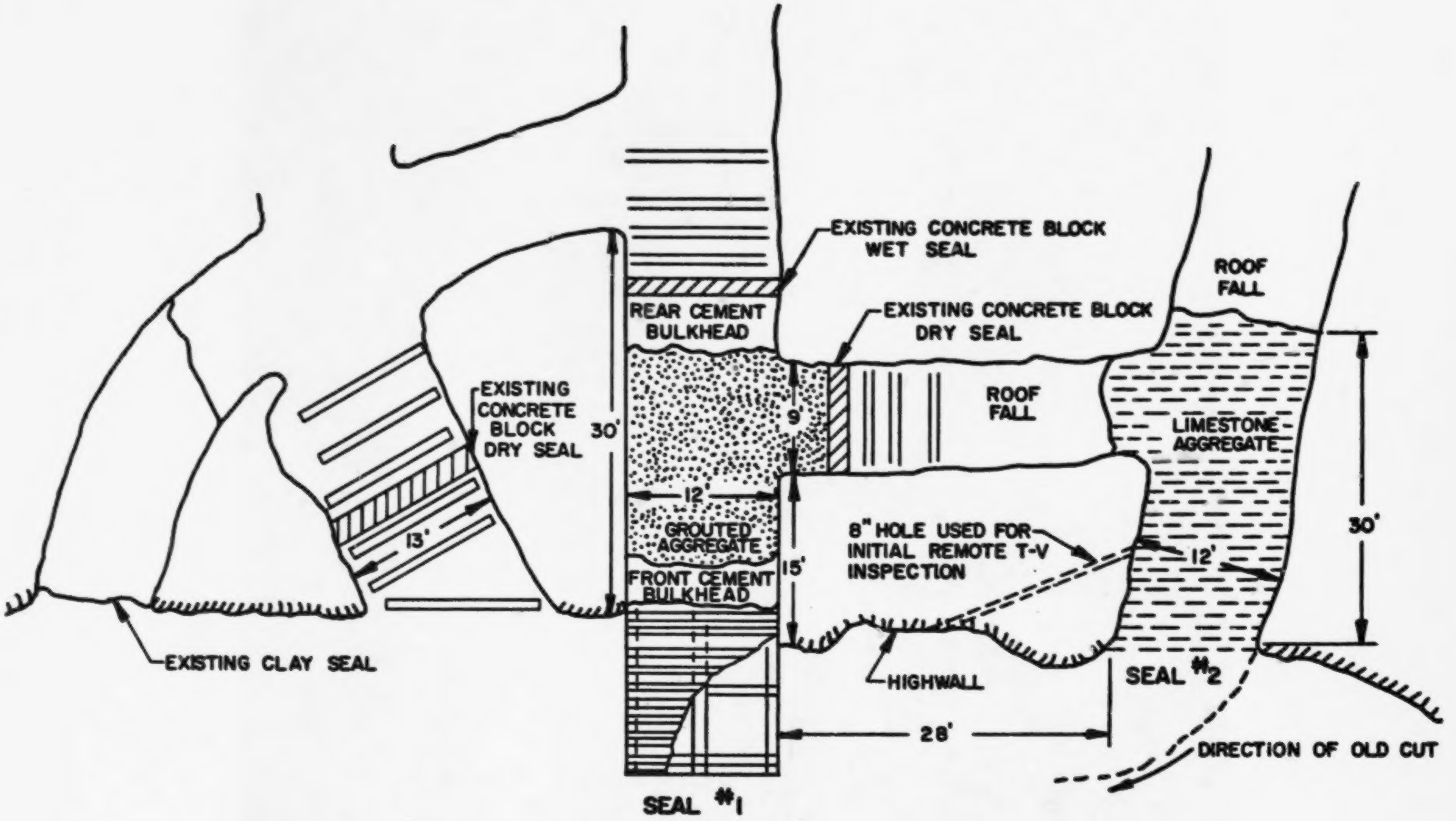


FIGURE 51 - VIEW OF MINE NO. RT5-2


EXISTING CONCRETE BLOCK WET SEAL
EXISTING CONCRETE BLOCK DRY SEAL
ROOF FALL
ROOF FALL
LIMESTONE AGGREGATE
REAR CEMENT BULKHEAD
EXISTING CONCRETE BLOCK DRY SEAL
30'
9'
12
GROUTED AGGREGATE
FRONT CEMENT BULKHEAD
15'
8" HOLE USED FOR INITIAL REMOTE T-V INSPECTION
12'
30'
13'
EXISTING CLAY SEAL
HIGHWALL
SEAL #2
28'
DIRECTION OF OLD CUT
SEAL #1

PLAN VIEW OF REMEDIAL CONSTRUCTION - MINE NO. RT5-2

FIGURE 52

footing for the dam and wall, was broken out to permit the installation of a drain line on the floor of the mine. To achieve a valid test, additional openings were made through the wet seal, allowing the mine water direct access to the new seal as the mine is flooded.

It was not necessary to remove the concrete block wet seal wall since there was sufficient room between this wall and the crosscut to obtain a seal on the drift wall, roof and floor with the rear bulkhead. The rear bulkhead was built in front of the wet seal wall and was self-supporting.

Additional preparation included thoroughly washing the floor and walls of the drift with a high-volume pump to aid in the bonding of the cement material to the wall and floor of the mine.

A drain line of 4-1/2-inch OD Fibercast pipe, 40 feet in length, was installed and extended 5 feet past the concrete block wall of the wet seal. This line was placed on a 6-inch bed of AASHO No. 67 crushed limestone which raised the pipe above the floor of the mine to insure a seal around the pipe. A screen analysis of this limestone is included in the Appendix in Table 28. The drain pipe was also covered with Halliburton Casing Kote. This is a bonded layer of angular sand grains around the pipe to give better hydraulic sealing characteristics to the drain pipe surface.

The mine seal was constructed in four stages. The first stage consisted of constructing a rear bulkhead of quick-setting, self-supporting cementitious material just in front of the concrete block wall of the wet seal.

The second stage consisted of the placement of six 2-inch grout pipes and then filling the area in front of the rear bulkhead and into the crosscut to the dry seal with AASHO No. 67 crushed limestone.

The third stage was the construction of the front bulkhead of the same material as the rear bulkhead. This bulkhead was built against the slope of the limestone aggregate to contain the grouting fluid used in the aggregate consolidation.

The final stage was the consolidation of the limestone aggregate with a grouting fluid to make the material impervious between the two bulkheads.

The process and materials used to build the rear and front bulkheads were the same as used to build three previous bulkheads in Mine No. 62-008. This work was recorded in a previous section of this report. In brief, this process involves preparing two slurries separately and mixing them together as they are being pumped into the mine. The slurries react to give a viscous quick-setting material which is able to support its own weight as it builds.

The pumping and mixing were accomplished using the pumps on a mobile pumping unit. As the materials were mixed, each slurry was placed in one of the two trailer-mounted pressure tanks.

From these tanks, the slurries were pumped simultaneously through separate lines to a common point at a wye connection. The resultant slurry then passed through a special mixing section of line as the reaction continued. This slurry was discharged through a nozzle designed for this particular application. The piping and mixing sections were mounted on a two-wheel dolly so the placement of the material could be controlled by an operator standing at or near the entrance to the mine.

The bulkhead was then built as previously described. It contained approximately 14 cubic yards of sodium silicate cement slurry. The angle of repose on the front and rear sides of the bulkhead was approximately 55 to 60 degrees.

After placement of the rear bulkhead, the slurry tanks used were cleaned to prepare them for containing the AASHO No. 67 aggregate (West Virginia No. 7) which was to be placed pneumatically into the mine. The tanks were loaded from dump trucks with a belt conveyor.

Simultaneous to the preparations and loading of the aggregate, grout pipes were installed in the mine drift in order to permit grouting of the limestone. All pipes were covered with Halliburton Casing Kote at the contact area through the front bulkhead. Three pipes of PVC plastic were installed on each side of the drift, one near the floor, one at the roof timbers and one about midway between the two. The pipes were nominal 2-inch size fitted with 2-inch plastic ball valves on the outer end. The pipes at the roof were 10.5 feet long. The center pipes were approximately 20 feet, and the lower pipe was 15 feet on the left-hand side and sufficiently long on the right-hand side to go back to the crosscut and bend into the crosscut near the dry seal. An extra pipe to be used as a hydrostatic measure was installed at the center of the opening along the roof line. It was not fitted with a valve, but had a 90-degree ell and a riser pipe for measuring the head during the grouting operation. The grout pipe arrangement is shown in Figure 53.

The dolly to be used for placement of the limestone in the mine was basically the same as that used for building the cement bulkhead, except that the 2-inch mixing pipe from that operation was replaced with a 4-1/2-inch OD aluminum pipe with a special 45-degree deflector on the end of the pipe.

The pneumatic placement of this limestone was accomplished by placing air on the pressure tanks containing the aggregate and using a special air manifold on the bottom of the tanks in the same manner as described previously. The material was moved with air through a 4-inch hose and the pipe on the dolly into proper placement in the mine drift. During this operation, 92,200 pounds of AASHO No. 67 aggregate were placed in the mine drift. This conveying was done in 148 minutes for an average rate of 622 pounds per minute while operating. The overall time for preparation, placing of the grouting pipes, loading, conveying and demobilizing was 11-1/2 hours.

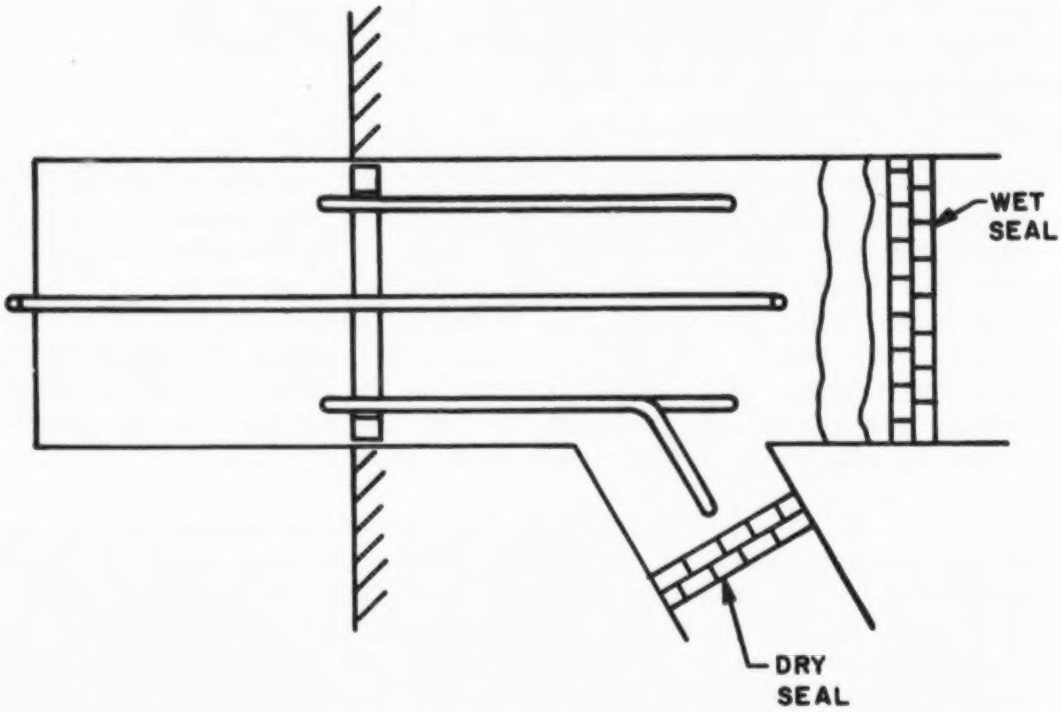
The next stage was the construction of a quick-setting cement bulkhead on the front slope of the aggregate which had been conveyed into the mine drift. Figure 54 shows the beginning of this operation. The materials used here were the same as those used in building the rear bulkhead. They were prepared and placed in a similar manner. Figure 55 shows the completed front bulkhead. About 9 cubic yards of slurry were used. The grout pipes with valves attached can be seen on either side. The hydrostatic pipe is evident coming from the top center portion of the bulkhead. The drain pipe can be seen on the floor of the mine in the lower left side.

On the following day, the grouting operation was performed using the same pumping equipment which had been used in building the bulkheads. The grouting material used was Halliburton LIGHT Cement. This material was a mixture of 65% portland cement and 35% fly ash, 6% bentonite and 9.9 gallons of water per bulk cubic foot of material, or 61 parts cement, 26 parts fly ash, 5.2 parts bentonite and 82.5 parts water by weight. This resulted in a slurry which weighed 12.7 pounds per gallon. The grouting slurry was pumped into the lower line on the right and left side simultaneously, at an average injection pressure of 10 psi. The valves on the upper lines were left open until the grout slurry flowed from them, at which time they were closed and the grouting was continued. The grout injection was moved progressively to higher grout pipes as the work proceeded.

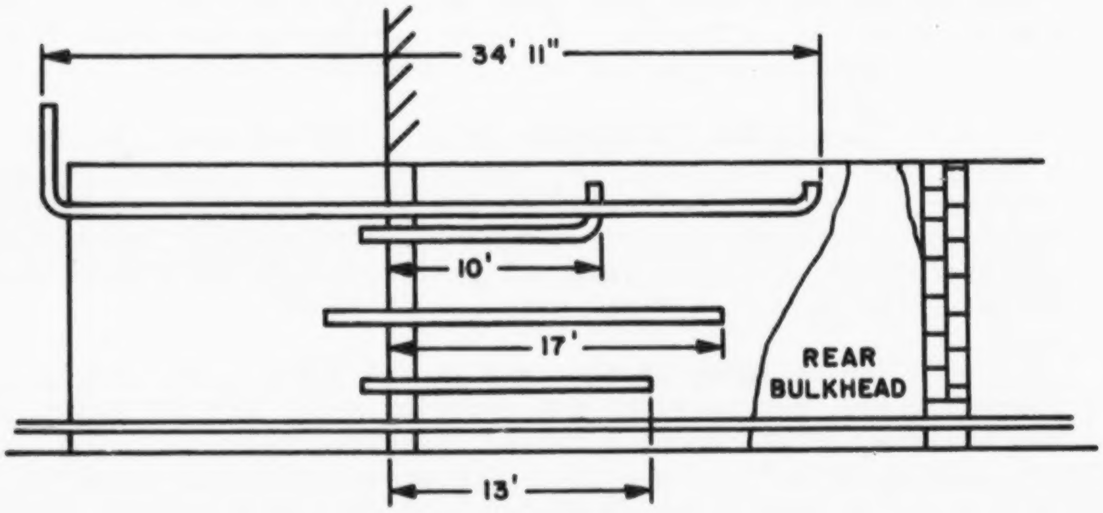
Pumping was continuous until a fill-up was obtained to within a few inches of the roof. At that time, leaks appeared around the top corners of the front bulkhead, so the slurry was altered by adding shredded cellophane, sawdust and shredded cane fiber. This material successfully closed the leaks at the top of the bulkhead and the grouting was concluded at a final pressure of 15 psi.

The building of the seal in the drift mine, consisting of the rear bulkhead, placement of the grouting pipes and the aggregate, and the building of the front bulkhead, required two days. The grouting required about half of one day. The preparation had required several days prior to this. The completed seal was approximately 25 feet in length and is shown in detail on the drawing attached as Figure 56. The total cost for this seal, including site preparation cost of $1,079.00, was $9,463.00 for materials and equipment.

A valve was installed on the drain line and closed 48 hours after the final grouting to commence flooding of the mine. Monitoring of the fluid level behind the seal then began (Table 29 - Appendix). At the time of closing the valve, the flow rate coming from the mine was 58.5 gallons per minute.


WET SEAL
DRY SEAL

TOP


34' 11"
10'
17'
REAR BULKHEAD
13'

GROUT PIPE ARRANGEMENT
SEAL NO. 1 - MINE NO. RT5-2
FIGURE 53

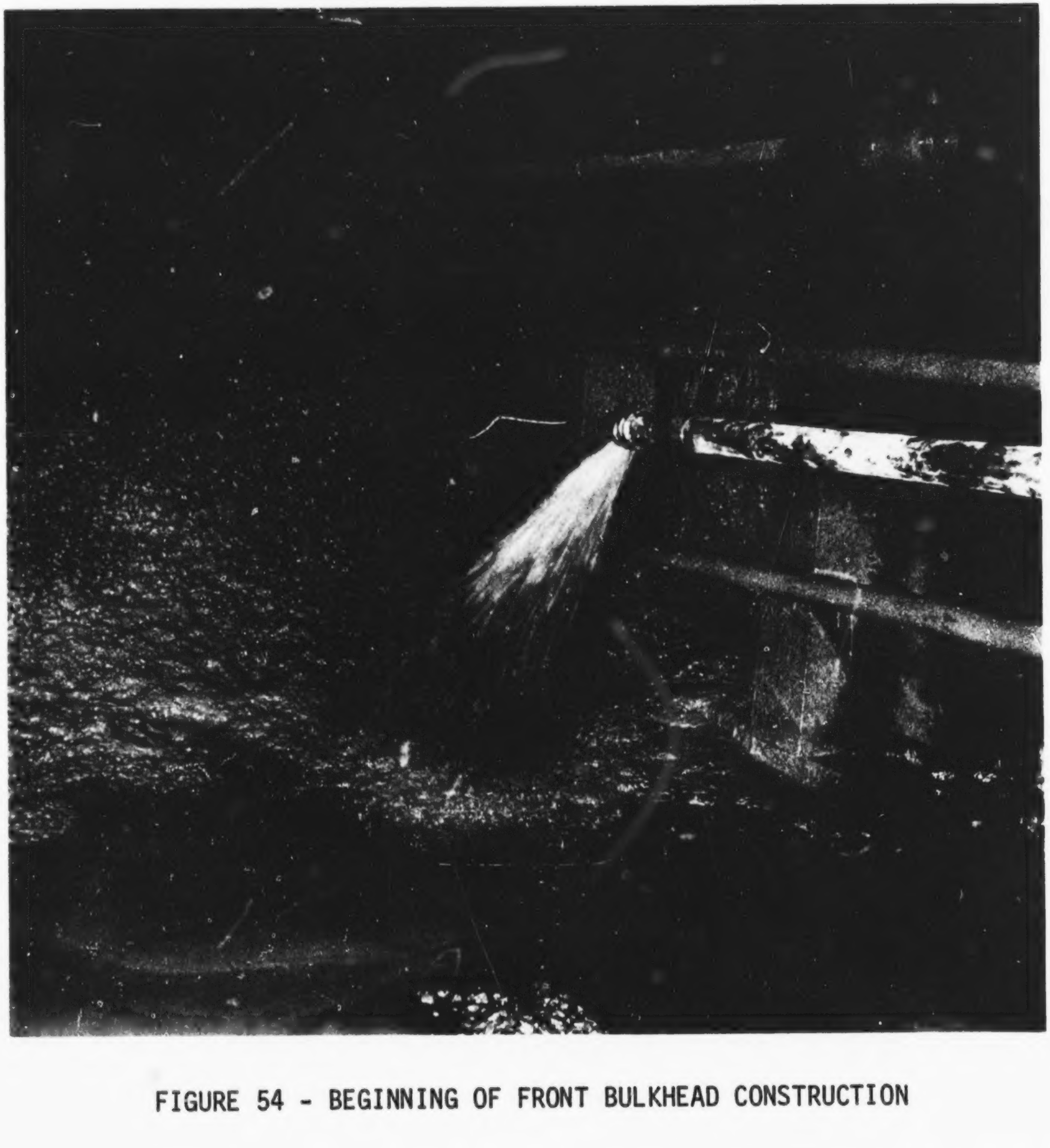


FIGURE 54 - BEGINNING OF FRONT BULKHEAD CONSTRUCTION

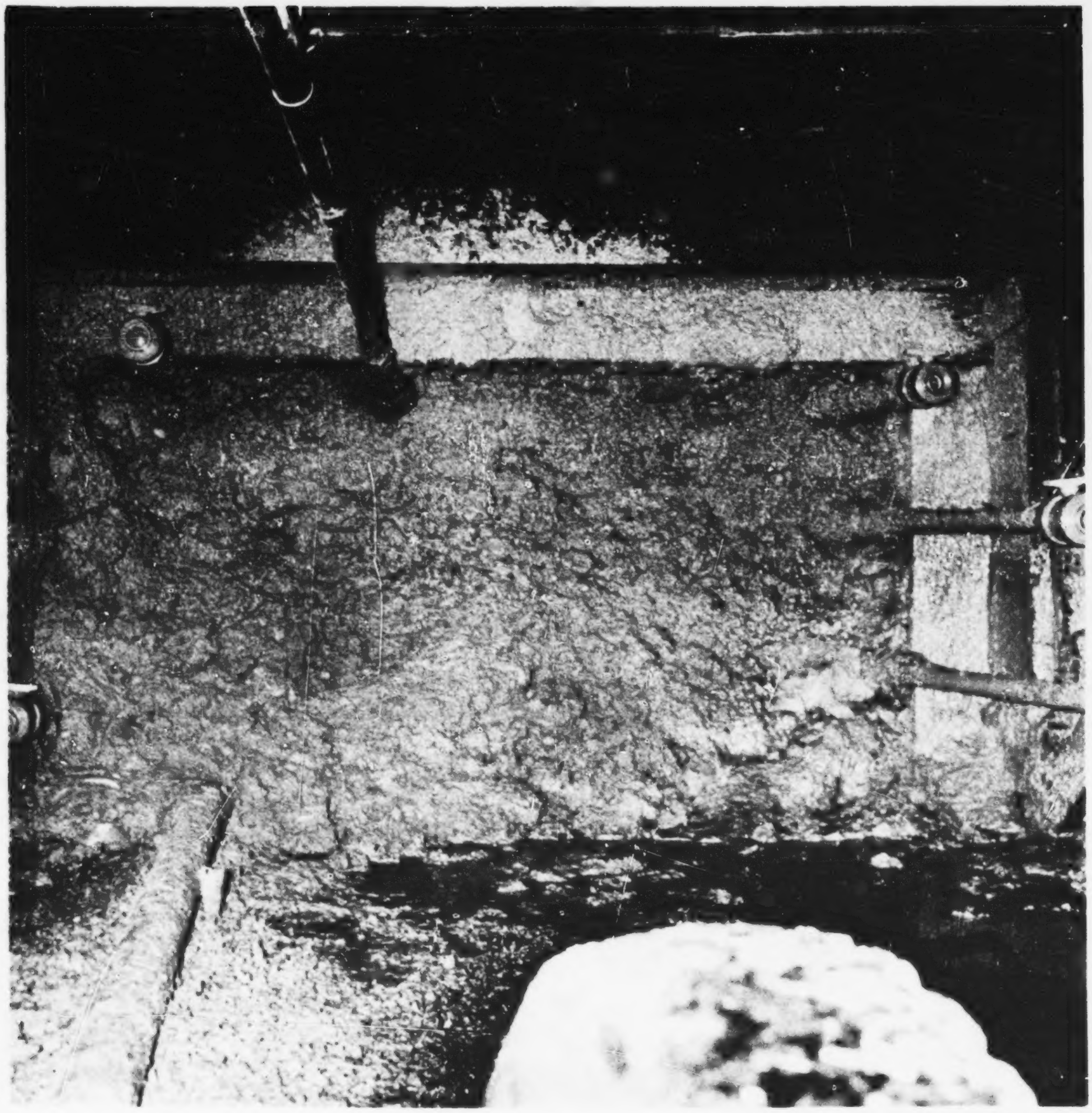
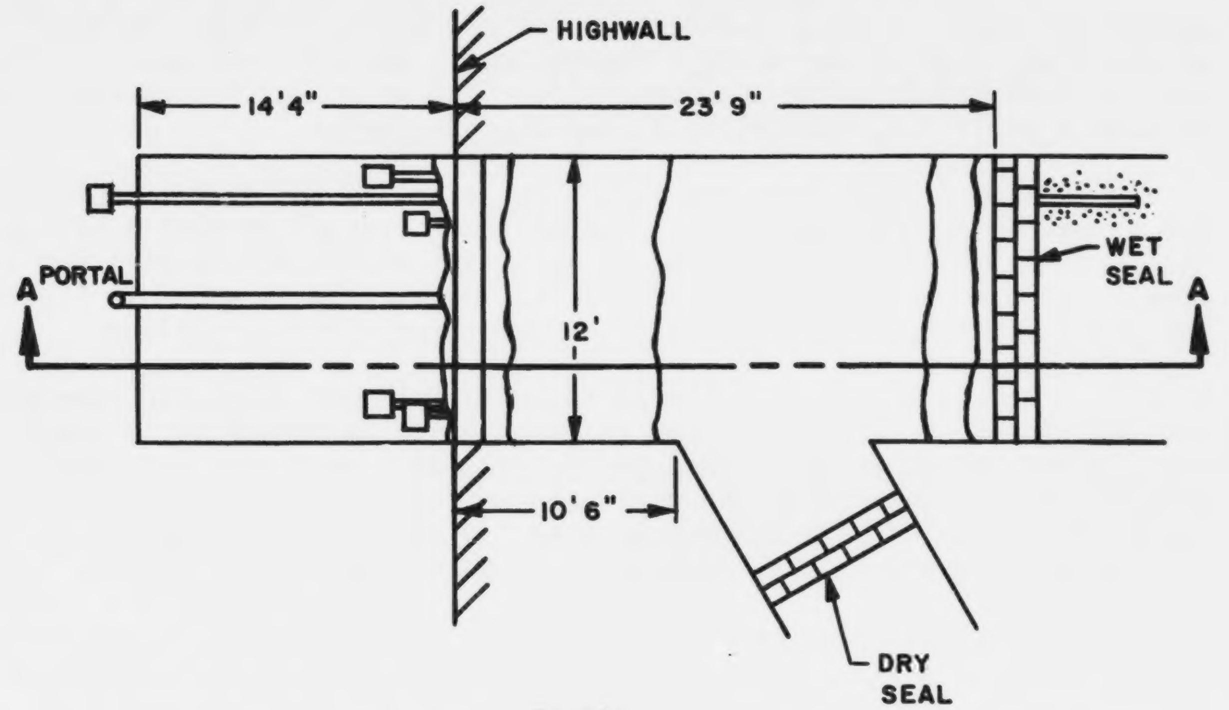
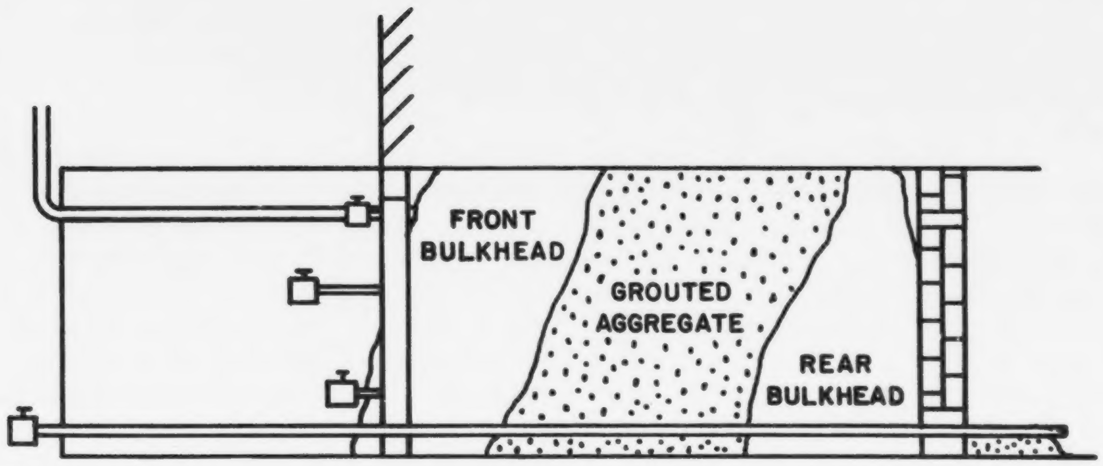


FIGURE 55 - COMPLETED FRONT BULKHEAD - MINE NO. RT5-2


HIGHWALL
14'4"
23'9"
A
PORTAL
12'
WET SEAL
A
10'6"
DRY SEAL

PLAN


FRONT BULKHEAD
GROUTED AGGREGATE
REAR BULKHEAD

SECTION A-A

CONSTRUCTION DETAIL
SEAL NO. 1 - MINE NO. RT5-2
FIGURE 56

On the sixth day after monitoring began, the head was 2.92 feet behind the seal. A small leakage was noticed coming through the backfill at the right side of the drift. The following day the head was 3.22 feet and the leak had increased substantially. The water was tested and found to have a pH of 2.8, indicating it was mine drainage.

Hand digging into the back slope right of the portal confirmed the direction of the water source. Additional excavation with a backhoe disclosed an opening about 6 feet to the right of the main portal and the remains of a masonry seal. A view of this excavation is shown in Figure 57. The drain valve was then reopened to prevent further head buildup.

It was decided to attempt to drill into the suspected opening and use closed circuit television to inspect it. Accordingly, a space was cleared for positioning an augering machine. An 8-inch hole was drilled in the top part of the coal vein as shown on Figure 52. At a depth of 18 feet, a void was encountered. A length of 7.5-inch OD casing was shoved into the hole to prepare for use of a television camera.

A television camera was pushed into the casing until the front of the camera was flush with the void. An attempt was made to identify the void which had been encountered through the picture received from the television camera. It was very difficult to determine exactly what was being shown on the screen. It appeared that either the lighting was inadequate or the scope of the camera was limited to the portion immediately in front of the camera. In looking at the dimly lighted picture on the television screen, something could be seen which appeared to be a pile of debris or a roof fall. Other than this identification, it was not possible to determine any other usable information.

It was then decided to continue excavation to remove additional material and completely expose the opening that had been encountered. This opening, designated as Opening No. 2, was found to be 28 feet to the right of Opening No. 1. After cleaning the front side, the opening was determined to be 12 feet wide. It was filled to the top of the coal with debris from the roof fall, leaving only 2 to 4 feet of void space between the top of the debris and the new roof. The existing roof was of a fossiliferous shale and appeared to be in good condition. A view of this exposed opening is shown in Figure 58.

After looking at the opening as it now appeared, it could be seen that the view given on the television camera was accurate and that the camera was located so that it was looking across the top of a large roof fall. The right-hand wall of this opening was diagonally about 15 feet in front of the camera. The existing mine roof after the fall was approximately 4 feet above the horizontal line of the camera for the entire distance to the opposite wall. Without seeing the exposed opening in full, it was difficult to interpret the information shown by the television camera. It is assumed that this method of viewing mine drifts could be useful, but only after additional work was done to make the lighting system and lens adaptable to this type service.

Since the opening was large and the roof fall continuous along the drift, a backhoe was used to remove the roof fall. As the clean-out progressed, protective timbering was installed using posts of 5-inch by 7-inch green oak timber on each side and cross members at the roof. Five of these were spaced 2 to 4 feet apart and extended into the opening for 12 feet.

As the cleaning progressed, water flow from Opening No. 2 increased with a corresponding reduction of the flow from the drain line of Opening No. 1. The opening was cleaned to a distance of 30 feet from the face. Very large boulders were encountered and had to be dragged out by chain.

Since the floor of the mine was lower than the ground outside the mine, a pump was used to keep the water pumped from the mine. The water flow increased as the clean-out progressed, so a second pump was installed. When the opening was sufficiently cleaned and the water level lowered, it was found that the main flow of water came from a point 20 feet back in the opening at the center between the two walls. Digging and probing disclosed a 15-inch ID drain line of vitrified clay laid in a ditch cut in the floor of the mine. The ditch, about 3 feet wide by 2 feet deep, was cleaned out by hand to the front of the mine. Most of the water coming from Opening No. 2 was coming through this drain tile which continued back into the mine under the roof fall for an unknown distance.

A plug was fabricated for the 15-inch drain tile from discs of 3/4-inch plywood and 3/8-inch gasket material, spaced 18 inches apart with 2 by 4-inch wood spacers. A 4-inch plastic pipe was placed through the plug. This pipe was then extended outside the mine so that it would serve as a drain line. The plug was inserted into the drain tile causing the flow to be diverted into the 4-inch drain line and out of the mine.

The special plug was grouted into the 15-inch drain tile by filling the ditch with cement grout slurry. The 4-inch drain line was brought out of the ditch on a slope so that the outer end was exposed outside the mine.

It was decided to place a limestone aggregate seal in the second opening. This seal was to be grouted along the top to help insure a seal across the roof section. It was suggested that agricultural lime be blended with the aggregate, if preliminary tests indicated that this was practical, in order to reduce the permeability for longer retention time and give increased surface area to cause faster neutralization.

A test was made at the laboratory facilities to observe the results of conveying a blended aggregate-lime mixture. A total of 10,000 pounds of 3/8-inch limestone chips containing approximately 15% agricultural lime was conveyed pneumatically from a 220-cubic foot pressure tank through 54 feet of 4-inch line. An initial pressure of 10 psi was placed on the tank containing the aggregate using a 110 scfm compressor. Then air from a 428 SCFM compressor was introduced into the bottom discharge manifold of the tank at a pressure of 30 psi. The aggregate moved steadily through the line and was discharged in an open area against a board wall. Extreme dusting was encountered which would be a problem when placing the material in a mine drift. A side view of the aggregate being discharged, shown in Figure 59, emphasizes the dust problem.

An excellent rate of 1,207 pounds per minute was obtained for the single tank load tested, but some means of alleviating the dust would be desirable for field application.

Preparation was made to use this type aggregate blend for a seal installation in Opening No. 2. Grout lines of 2-inch plastic pipe were installed in the opening as shown in Figure 60. The purpose was to grout only the upper part of the aggregate to insure a seal against the roof of the mine in the event any settling should take place. Pipes were placed along both sides of the opening at the roof timbers, in the center of the vertical opening and on the floor.

The opening was filled with a large quantity of AASHO No. 8 crushed limestone blended with approximately 15% agricultural lime. Screen analysis for AASHO No. 8 (West Virginia No. 12) aggregate is in Table 28 of the Appendix. The aggregate seal was placed against a roof fall in the drift about 30 feet from the opening at the highwall. The agricultural lime consisted primarily of the fine limestone dust that is obtained when limestone is crushed. The limestone was stockpiled on the ground and then loaded into a 500 cubic foot pneumatic trailer using a highlift and belt conveyor. The agricultural lime was in 50-pound bags and was blended with the crushed limestone on the belt conveyor as it was being put into the pneumatic tank.

The rock and lime mixture was conveyed from the pneumatic tank into the mine drift through 40 feet of 4-inch rubber hose and the 20 feet of 4-inch pipe contained on the placement dolly. This was the same equipment used for placement of the rock in Opening No. 1.

One problem which was prevalent in this test was the extreme amount of dust created when the limestone was blown into the mine drift. In order to combat this problem, the ratio of lime dust was reduced to 10% and a pump was used to spray a one-half inch jet of water against the deflector plate on the discharge end of the dolly at the same time as the limestone was impinging on the plate. This helped to some degree, but even using this water spray the dust problem was still bad.

A total of 330,000 pounds of material was pneumatically placed in the mine in a net time of 346 minutes, for an average rate of 955 pounds per minute. The overall time for the placement was 23 hours, including all the downtime, loading time, etc., involved in the operation.

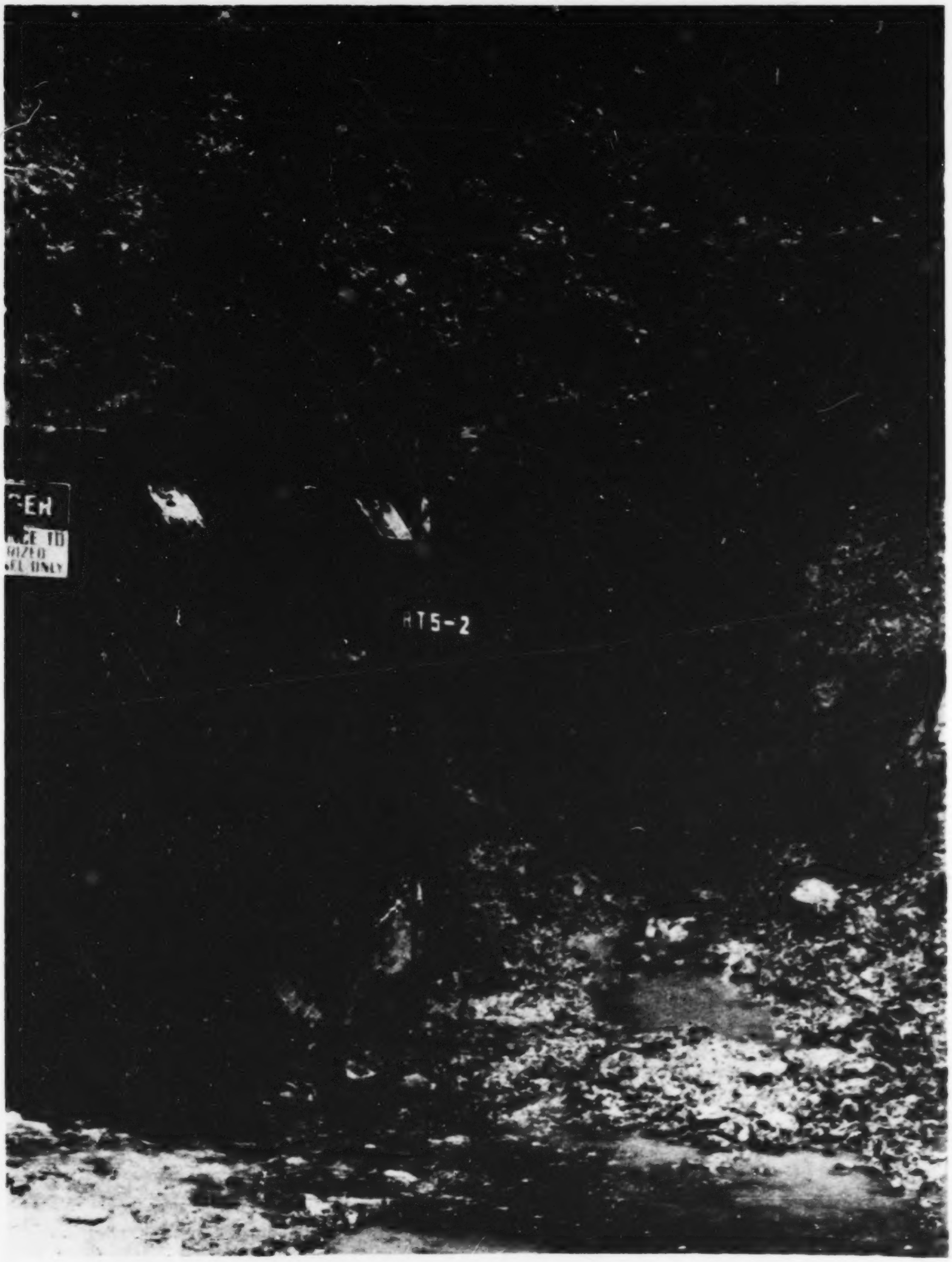


FIGURE 57 - OPENING EXCAVATED RIGHT OF MAIN PORTAL - MINE NO. RT5-2

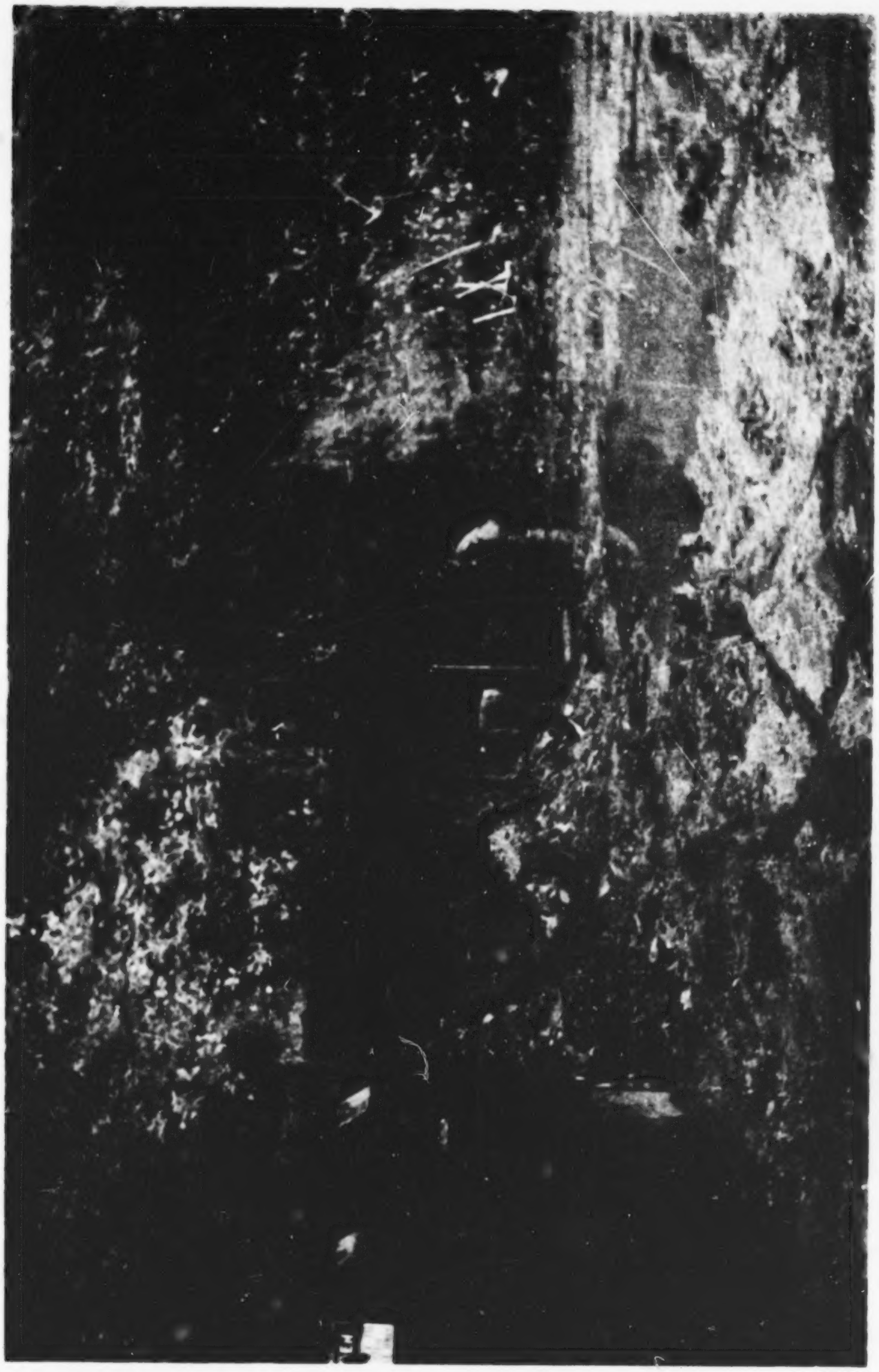


FIGURE 58 - OPENING NO. 2 AFTER EXPOSING FRONT - MINE NO. RT5-2

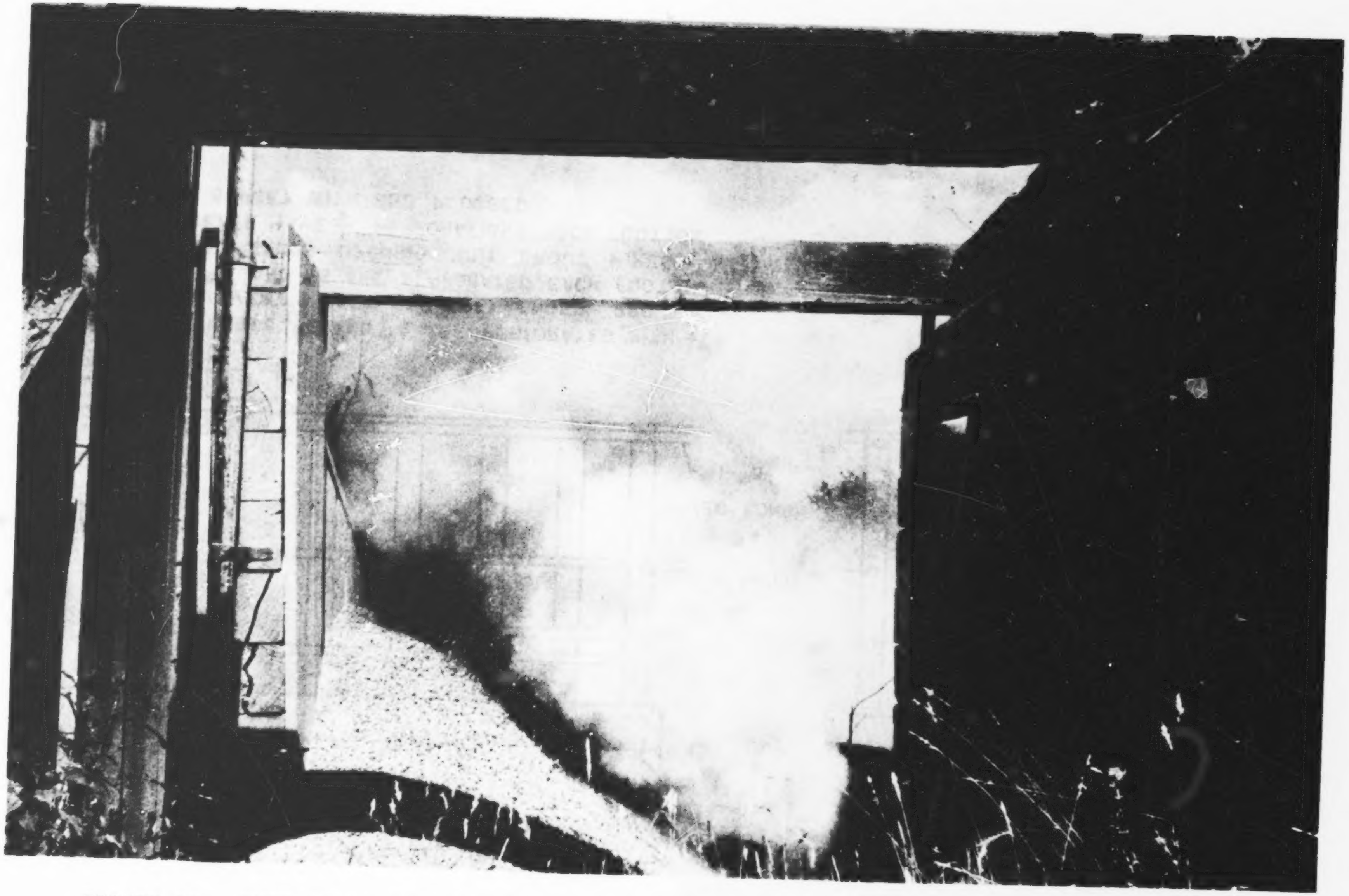
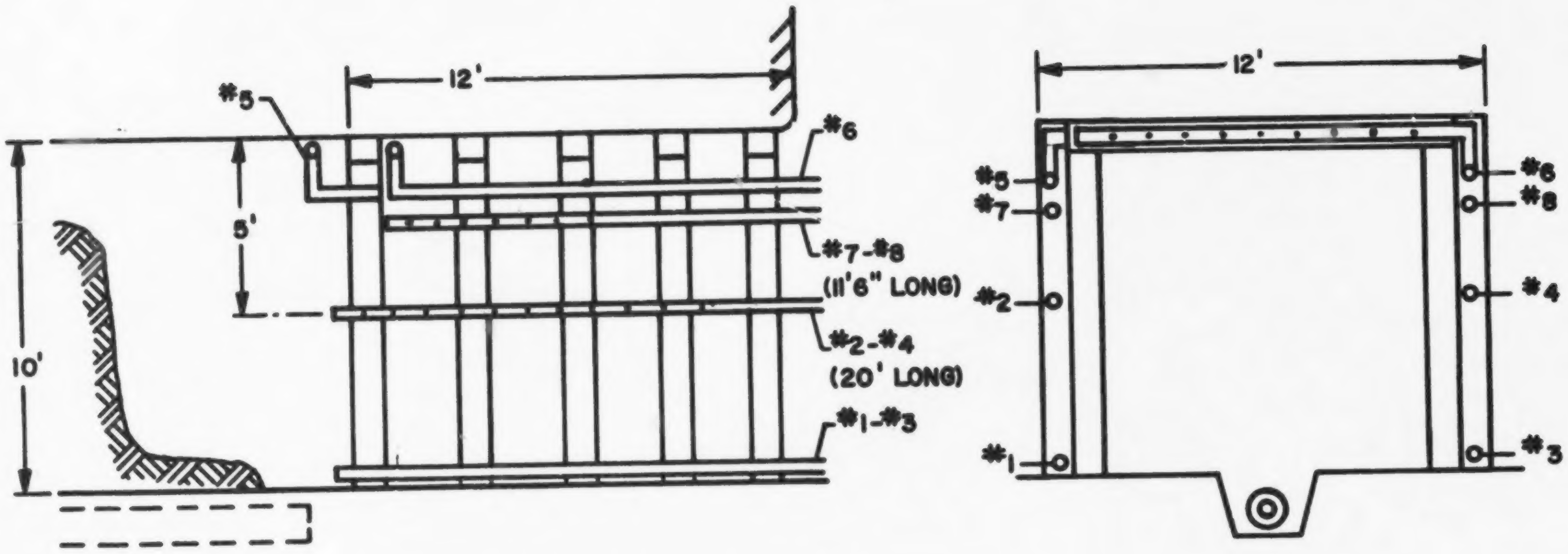


FIGURE 59 - DUST CREATED WITH PNEUMATIC CONVEYING OF AGGREGATE CONTAINING AGRICULTURAL LIME


12'
#5
#6
5'
10'
#7-#8
(11'6" LONG)
#2-#4
(20' LONG)
#1-#3
12'
#5
#7
#2
#1
#6
#8
#4
#3

NOTE: GROUT PIPES #2 AND #4 ARE PERFORATED WITH $\frac{1}{2}$" HOLES EVERY 3 FEET AND OPEN ENDED. GROUT PIPES #5 AND #6 ARE PERFORATED EACH FOOT ON PORTION ACROSS OPENING ONLY. GROUT PIPES #7 AND #8 HAVE $\frac{1}{2}$" PERFORATIONS EACH FOOT ON INNER 5 FEET WITH END PLUGGED.

GROUT PIPE PLACEMENT - OPENING NO. 2
FIGURE 60

This work was spaced over 2 days' working time.

After the material was all placed in the drift opening, a valve was installed on the 4-inch drain line and a vertical length of clear tubing was provided to measure the water head behind the limestone in the mine.

The grouting of the upper portion of the limestone plug was accomplished by pumping a Halliburton LIGHT Cement slurry as a grouting slurry. This was the same slurry that was used to grout the aggregate in Opening No. 1 of this mine. The grout was pumped into the two upper pipes on either side of the opening. The grouting was accomplished by pumping approximately 100 cubic feet of grout slurry into the upper portion of the limestone.

Figure 61 shows a section view of the seal which was installed in Opening No. 2 of Mine RT5-2. The aggregate seal is approximately 30 feet in length.

Since the area immediately in front of the mine was low and filled readily with water that came through the limestone, a retaining wall was built in front of the opening with a 90-degree weir installed to measure the flow rate. A view of the completed installation at Opening No. 2 is shown in Figure 62.

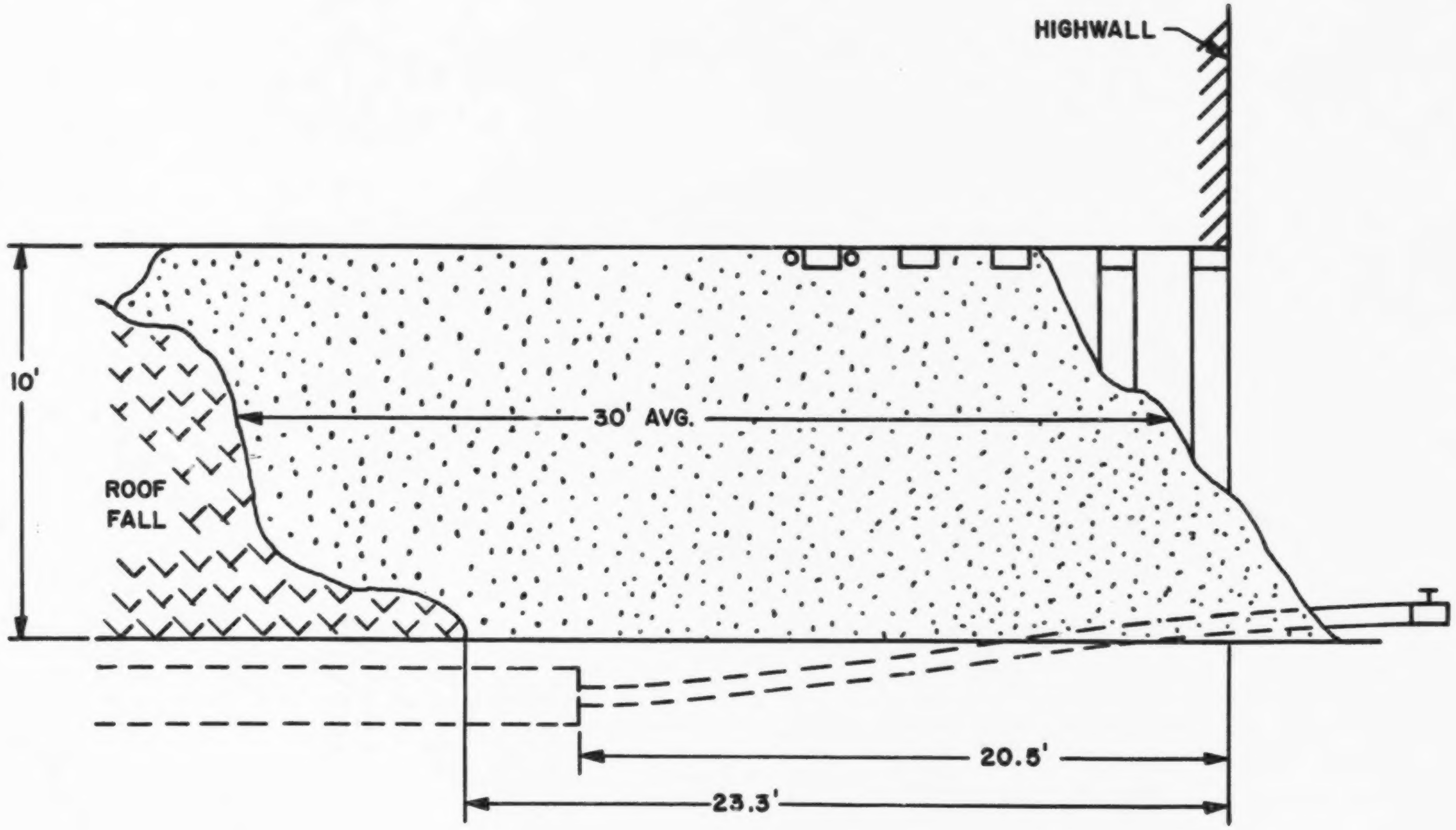
The graded aggregate seal placed in Opening No. 2 was constructed at a cost of $8,463.00. This cost included $3,447.00 for site preparation, $1,696.00 for materials and $3,320.00 for equipment and operators. This cost was higher than a previous seal of this type because of the excessive excavation required to prepare the opening, the extra materials required for grouting the upper section and a corresponding increase in the necessary equipment.

After the seal was placed, monitoring was resumed for Opening No. 1 and initiated for Opening No. 2. The head of water impounded and the flow through the aggregate seal of Opening No. 2 was recorded (Table 30 - Appendix). The initial flow of 25 gpm with a head of 3.3 feet of impounded water diminished to a flow of 3.6 gpm with a head of 5.44 feet in about 24 days. Since the field work on this project ceased about 30 days after the seal was installed, the monitoring data stopped at this point. However, monitoring was continued by Federal Water Pollution Control Administration personnel from the Norton, West Virginia Acid Mine Drainage Laboratory.

The water impounded behind the seals in the two openings had a pH of 2.9 to 3.2, an acidity of 512 to 765 mg/l, an iron content of 104 to 282 mg/l and sulfate of 868 to 1,152 mg/l. After passing through the limestone seal, the water neutralized to a pH of 5.6 to 6.3, an acidity of 0 to 103 mg/l, iron content of 36 to 57 mg/l, and sulfate of 780 to 940 mg/l. The hardness did increase as a result of the lime treatment, however (Table 31 - Appendix).

It was impossible to tell whether the leakage through the limestone seal was channeled through one area or across the entire seal because of the condition in front of the opening. This area was low and water remained impounded, covering the lower portion of the seal and obscuring observation of the leakage.

Additional information on the monitoring of the two openings after cessation of field work on the project has been furnished by the Norton FWPCA personnel. The head on the impounded water behind both seals has slowly increased, reaching 6.48 feet in Opening No. 2 and 4.74 feet at Opening No. 1 by December 30, 1969. The flow remains at 3.6 gpm through the limestone seal. Water quality likewise is similar to that reported in Table 31.


HIGHWALL
10'
30' AVG.
ROOF
FALL
20.5'
23.3'

SECTION VIEW OF SEAL - OPENING NO. 2
FIGURE 61

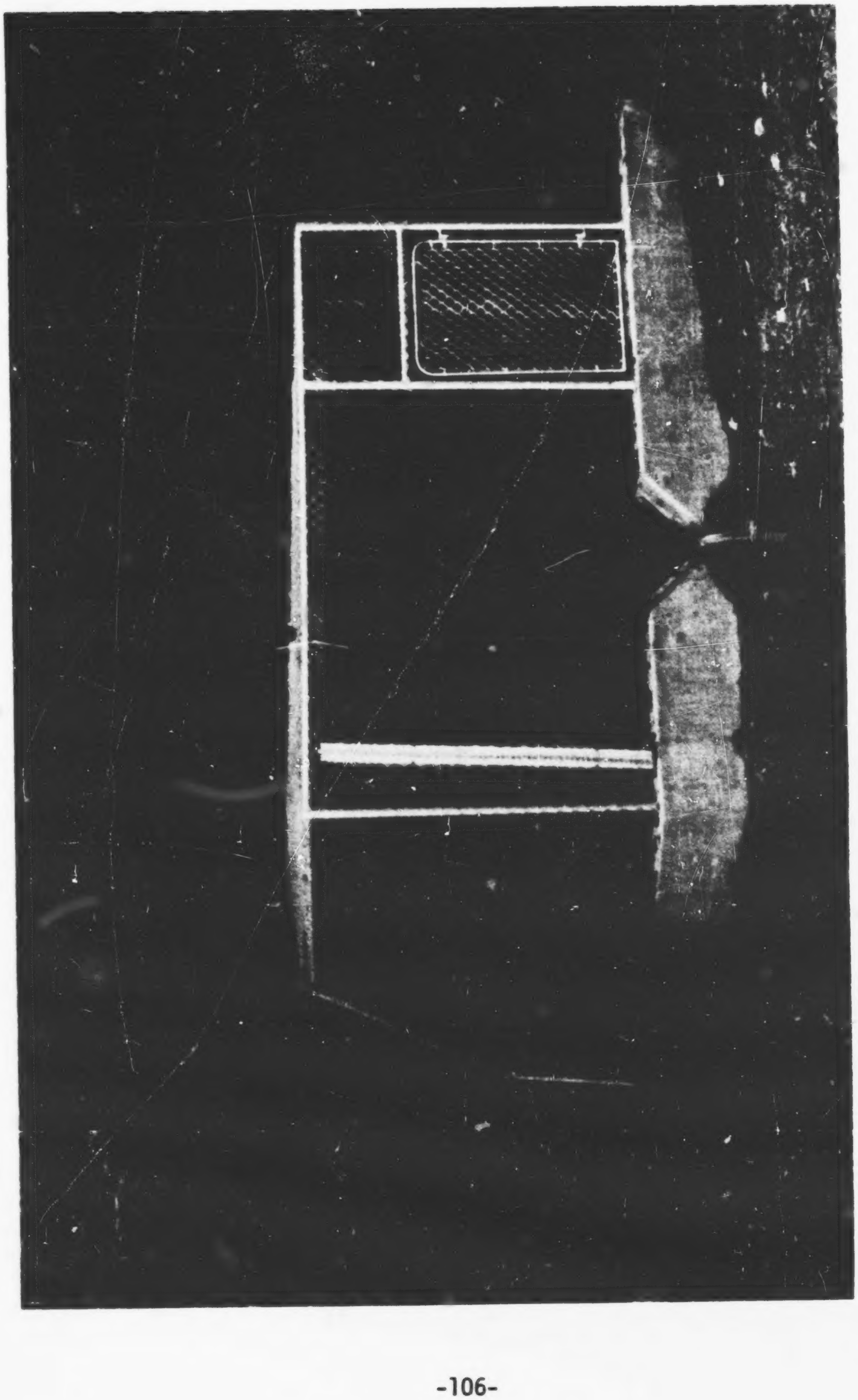


FIGURE 62 - COMPLETED INSTALLATION - OPENING NO. 2

ACKNOWLEDGEMENTS

In the development and accomplishment of the studies and investigation requisite to the preparation of this report, the valuable assistance and suggestions of the Project Officers are gratefully acknowledged. During the Contract, four different Project Officers were assigned. They are listed below in chronological order:

Dr. Edward J. Martin - Washington, D.C.
July 15, 1968 to August 27, 1968

Mr. John R. Hyland - Wheeling, West Virginia
August 28, 1968 to March 16, 1969

Dr. Donald L. Warner - Cincinnati, Ohio
March 17, 1969 to September 17, 1969

Mr. Donald J. O'Bryan - Washington, D.C.
September 18, 1969 to end of Contract

The analyses of the many water samples by the Federal Water Pollution Control Administration Laboratory in Wheeling, West Virginia were deeply appreciated.

Mr. Paul W. Hornor, Consulting Mining Engineer of Clarksburg, West Virginia, rendered valuable assistance in general consultation during the contract.

The cooperation of the personnel from the Federal Water Pollution Control Administration Laboratory at Norton, West Virginia is also gratefully acknowledged.

The helpful suggestions and comments from Mr. Ronald D. Hill, Robert A. Taft Water Research Center, Federal Water Pollution Control Administration, Cincinnati, Ohio were greatly appreciated.

REFERENCES

American Petroleum Institute, Publication RP 13B, November, 1962, 15p.

Halliburton Company, Part II, "Selection and Recommendation of Twenty Mine Sites", August 23, 1967, 161p.

Halliburton Company, Part IV - "Additional Laboratory and Field Tests for Evaluating and Improving Methods for Abating Mine Drainage Pollution", May 21, 1968, 134p.

PATENTS

Two patent applications have resulted from the work conducted on this contract. The description of these patents is given below:

(1) FWP-1531

E. L. Paramore et al

Filed: May 13, 1969

Serial No.: 824,159

For: "Forming Self-Supporting Barriers
in Mine Passages and the Like"

(2) FWP-1564

Tommy R. Gardner et al

Filed: July 9, 1969

Serial No.: 840,494

For: "Method of Plugging Mine Passages
Having Water Emanating Therefrom"

ABBREVIATIONS

alka.	alkalinity
AASHO	American Association of State Highway Officials
bpm	barrels per minute (1 barrel = 42 gallons)
cfs	cubic feet per second
cm	centimeters
Cr.	Creek
disch	discharge
ft	foot, feet
gpm	gallons per minute
hr	hour
hrs	hours
ID	inside diameter
in.	inch, inches
lbs/ft	pounds per foot
lbs/gal	pounds per gallon
lbs/min	pounds per minute
mg/l	milligrams per liter
ml	milliliters
No., #	number
OD	outside diameter
psi	pounds per square inch
scfm	standard cubic foot per minute
USGS	United States Geological Survey

APPENDIX

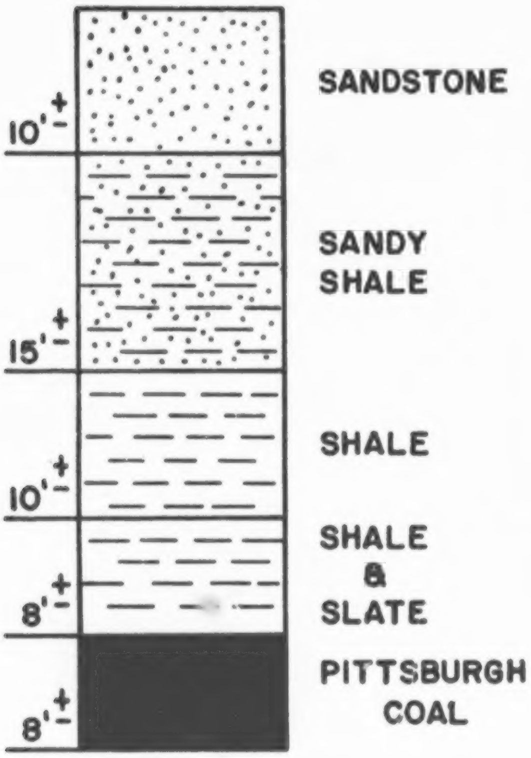

10'±
SANDSTONE
15'±
SANDY SHALE
10'±
SHALE
8'±
SHALE & SLATE
8'±
PITTSBURGH COAL

FIGURE 63-GEOLOGIC SECTION-MINE NO. 14-042A

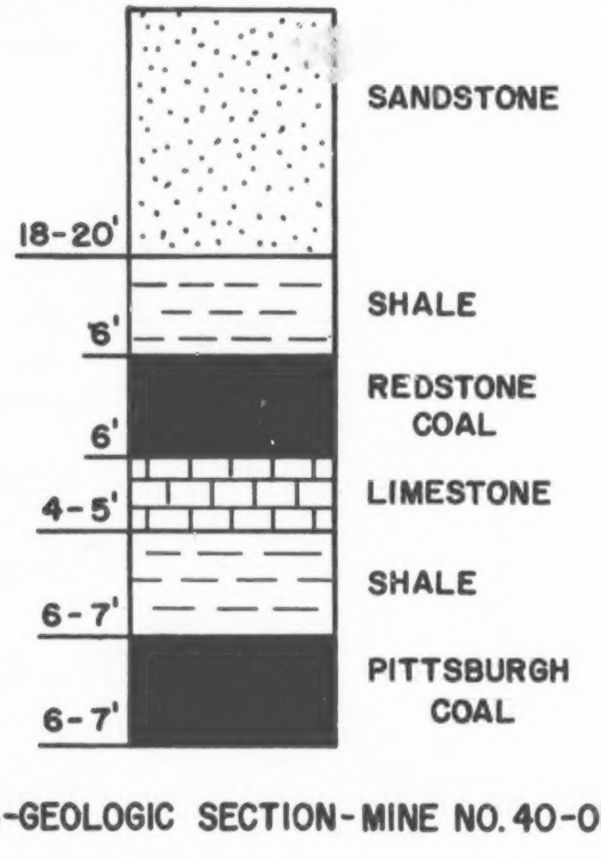

18-20'
SANDSTONE
6'
SHALE
6'
REDSTONE COAL
4-5'
LIMESTONE
6-7'
SHALE
6-7'
PITTSBURGH COAL

FIGURE 64-GEOLOGIC SECTION-MINE NO. 40-016

BLANK PAGE

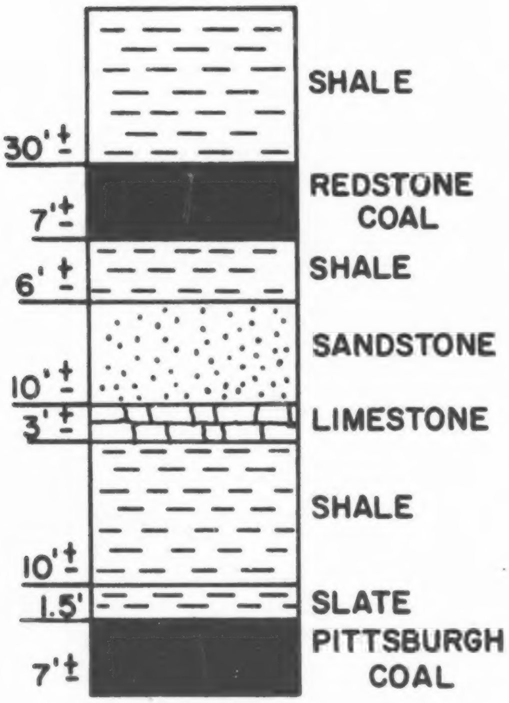

SHALE
30'±
REDSTONE COAL
7'±
SHALE
6'±
SANDSTONE
10'±
3'±
LIMESTONE
SHALE
10'±
1.5'
SLATE
PITTSBURGH COAL
7'±

FIGURE 65-GEOLOGIC SECTION-MINE NO. 62-008

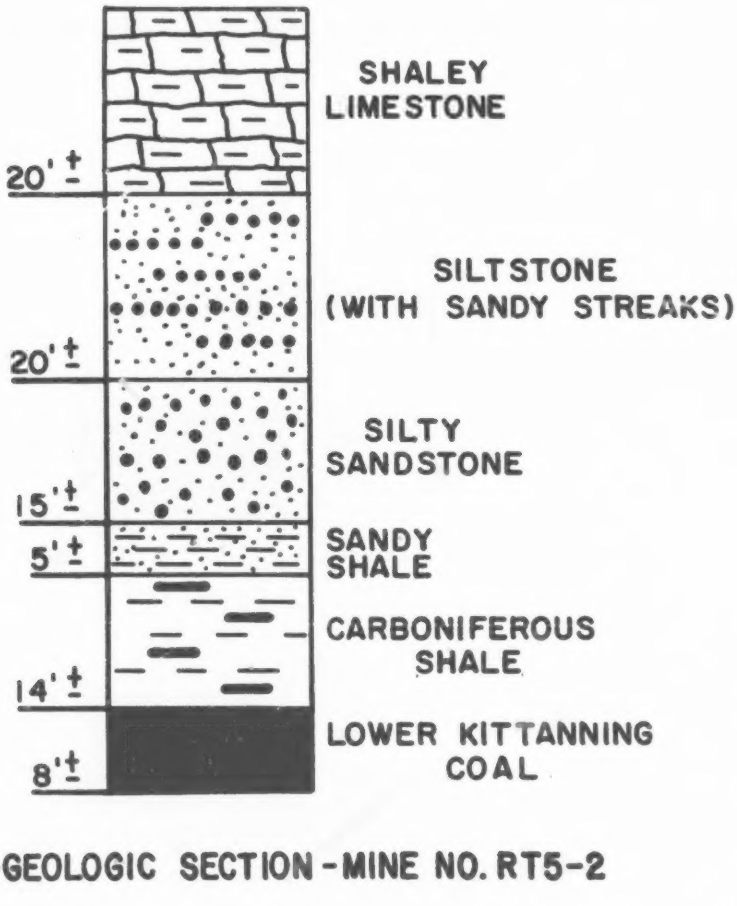

SHALEY LIMESTONE
20'±
SILTSTONE (WITH SANDY STREAKS)
20'±
SILTY SANDSTONE
15'±
SANDY SHALE
5'±
CARBONIFEROUS SHALE
14'±
LOWER KITTANNING COAL
8'±

FIGURE 66-GEOLOGIC SECTION-MINE NO. RT5-2

TABLE 1

FLUID PROPERTY TEST[1] - MINE NO. 14-042A

Gelled Fluid		Viscosity	Filter Loss[2]		Time After
Batch No.	Batch Size (gallons)	Marsh Funnel (seconds)	cc	Filter Cake (inches)	Mixing (hrs & mins)
1	820	39	19.1	2/32	:05
1	820	43	17.0	3/32	1:50
2	10,000	47	18.2	2/32	:05
2	10,000	58	17.2	2/32	1:00
2	10,000	63	16.8	2/32	1:30
3	10,300	56	16.6	3/32	:20
3	10,300	55	16.2	3/32	1:10
-[3]	-[3]	62	15.8	2/32	-
4	10,300	48	15.8	2/32	:25
4	10,300	47	15.8	2/32	1:10
5	10,300	45	16.4	2/32	:30
5	10,300	48	17.1	12/32	22:30

[1] Tests were conducted using standard API test procedures given in API Bulletin RP 13B.

[2] Filter loss measured in Baroid Filter Press using filter paper. Reading shown was measured at 30 minutes with 100 psi pressure.

[3] Sample taken from fluid inside mine about 2-1/2 feet below fluid level.

TABLE 2

MINE FLUID DATA - MINE NO. 14-042A

Date	Time	Gelled Fluid Added (gallons)	Fluid Level (ft)	Level Change (ft)	Discharge Rate (gpm)	Sample No.
9-30-68	7:00 a.m.	-	5.00	-	0.40	2
9-30-68	11:47 a.m.	800	5.10	+0.10	0.40	-
9-30-68	3:25 p.m.	9,400	5.52	+0.42	0.50	-
10- 1-68	7:30 a.m.	-	4.94	-0.58	0.40	3
10- 1-68	10:55 a.m.	10,300	5.69	+0.75	0.50	4
10- 2-68	7:30 a.m.	-	5.19	-0.50	0.50	5
10- 2-68	10:32 a.m.	10,300	6.67	+1.48	0.75	-
10- 2-68	3:45 p.m.	-	6.31	-0.35	0.55	-
10- 3-68	7:00 a.m.	-	6.15	-0.17	-	-
10- 3-68	12:45 p.m.	8,200	9.04	+2.90	-	-
10- 4-68	8:00 a.m.	-	7.81	-1.23	0.55	6
10- 4-68	11:03 a.m.	2,100	8.98	+1.17	0.50	-
10- 5-68	7:30 a.m.	-	8.83	-0.15	0.55	7
10- 6-68	10:30 a.m.	-	8.33	-0.50	0.55	8
10- 7-68	7:30 a.m.	-	8.17	-0.17	0.55	9
10- 8-68	10:00 a.m.	-	7.88	-0.29	0.45	10
10-10-68	3:30 p.m.	-	7.60	-0.27	0.60	11
10-12-68	3:30 p.m.	-	7.40	-0.21	0.50	-
10-17-68	-	-	6.96	-0.44	0.54	12
10-29-68	-	-	6.56	-0.40	0.48	13
11- 1-68	-	-	6.54	-0.02	-	-
11- 8-68	-	-	-	-	0.48	14

TABLE 3

WATER MONITORING DATA - MINE NO. 14-042A

Date	Sample No.	Fluid Level (feet)	Flow Rate (gpm)
11-21-68	15	6.40	0.48
11-26-68	16	6.36	0.50
12- 5-68	17	6.30	0.60
12-13-68	18	6.28	0.70
12-18-68	19	6.25	0.52
1- 2-69	20	6.20	0.80
1- 8-69	21	6.23	-
1- 9-69	22	-	1.05
1-16-69	23	6.20	0.95
1-24-69	24	6.17	1.25
1-30-69	25	6.02	1.80
2- 4-69	26	6.02	3.00
2-10-69	27	6.03	2.40
2-18-69	28	5.96	2.10
2-24-69	29	6.04	2.10
3- 3-69	30	6.01	1.38
3-10-69	31	6.04	1.10
3-17-69	32	6.01	0.95
3-24-69	33	6.00	1.10
4- 1-69	34	6.00	0.85
4- 7-69	35	6.01	1.40
4-14-69	36	-	1.55
4-21-69	37	6.02	1.50
4-28-69	38	6.00	1.10
5- 5-69	39	6.03	1.15
5-12-69	40	6.04	1.70
5-19-69	41	6.04	1.40
5-26-69	42	6.06	1.30
6- 2-69	43	6.08	1.10
6-10-69	44	6.08	0.95
6-16-69	45	6.08[1]	0.70
6-23-69	46	6.06	0.65
6-30-69	47	6.04	0.60
7- 7-69	48	6.04	0.60
7-21-69	49	6.02	0.65
7-28-69	50	6.00	0.62
8- 4-69	51	6.00	0.52
8-11-69	52	5.98	0.58
8-18-69	-	5.98	-
8-25-69	-	5.98	-
9- 2-69	-	5.96	-
9-22-69	-	5.92	0.55
10- 6-69	-	5.95	0.51
10-13-69	-	5.95	0.40

[1] Water-level recorder was still making a straight line after a hard rain. It was suspected that heavy gel around float was preventing float to fluctuate as water level varied.

TABLE 4

WATER ANALYSES - MINE NO. 14-042A

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
9-21-68	5966	2.8	4520	0	1620	1020	5850	340
9-30-68	6100	3.2	4650	0	1400	876	5460	404
10- 1-68	6366	3.5	4510	0	1470	912	4316	420
10- 1-68	5720	3.6	4920	0	1325	900	5590	334
10- 2-68	5834	3.7	4730	0	1270	960	6110	410
10- 4-68	5410	3.0	4480	0	1215	900	6110	480
10- 5-68	5480	3.0	4140	0	1415	1032	5330	880
10- 6-68	5580	3.0	3970	0	1300	852	5330	380
10- 7-68	5370	3.0	3985	0	1330	831	4810	304
10- 8-68	5460	3.1	4360	0	1030	792	4940	394
10-10-68	5240	3.2	4350	0	990	749	4550	334
10-17-68	5360	2.9	4340	0	1175	852	6500	224
10-29-68	5460	3.0	4180	0	1400	864	4550	304
11- 8-68	5540	2.8	3910	0	1400	840	5200	364
11-21-68	5660	3.2	4030	0	1250	936	5850	260
11-26-68	4750	3.2	4070	0	1370	480	6110	132
12-13-68	5440	2.8	4290	0	1065	696	5070	281
12-18-68	5442	3.0	4120	0	1160	840	4680	414
12-26-68	5280	3.0	3700	0	1570	840	5850	180
1- 2-69	5220	2.9	3700	0	1480	900	5850	205
1-16-69	5200	2.9	4200	0	1285	816	5590	284
1-24-69	4600	2.9	3875	0	955	832	5050	131
1-30-69	5020	2.9	3700	0	1330	701	3900	184
2- 4-69	4850	2.9	3670	0	510	692	4550	177
2-10-69	4400	2.8	3900	0	890	625	4345	185
2-18-69	4800	2.6	3620	0	1630	790	4400	225
2-24-69	4808	2.7	3640	0	1360	685	5100	250
3- 3-69	4880	3.1	3815	0	710	660	3600	115

TABLE 4 - Concluded

WATER ANALYSES - MINE NO. 14-042A

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
3-10-69	4600	3.0	3646	0	28	735	4200	160
3-17-69	4800	3.2	3420	0	1010	837	4400	164
3-24-69	4738	2.6	3000	0	575	631	3900	178
4- 2-69	4700	2.9	3335	0	960	653	4900	160
4- 7-69	4396	3.0	2820	0	940	587	4500	175
4-14-69	4600	2.6	2700	0	830	720	4000	167
4-21-69	4322	3.1	2975	0	940	646	4200	160
4-28-69	3940	3.1	2470	0	575	553	3200	145
5- 5-69	4580	1.9	3306	0	915	238	3300	161
5-12-69	4080	3.3	2560	0	100	522	3800	165
5-19-69	4268	2.6	2760	0	965	635	4200	158
5-26-69	3920	2.5	2800	0	1208	217	4200	152
6- 2-69	4300	2.8	2745	0	965	498	4000	156
6-10-69	4100	2.8	2725	0	1020	571	4100	146
6-16-69	4100	3.2	2550	0	940	44	3500	136
6-23-69	4240	3.1	2820	0	1220	475	4300	136
6-30-69	3734	3.1	2650	0	1140	622	3825	151
7- 7-69	4018	3.0	2765	0	1080	376	3100	153
7-21-69	3619	3.0	1285	0	1440	413	3105	143
7-28-69	3610	2.6	2250	0	1080	411	2700	126
8- 4-69	2582	2.9	2670	0	1150	538	2600	149
8-11-69	3140	2.6	2475	0	1085	476	3440	148

TABLE 5

WATER MONITORING DATA - MINE NO. 40-016

Date	Fluid Level - (inches) Opening #1	Opening #2	Flow Rate - (gpm) Opening #1	Opening #2	Remarks
9-20-68	-	100.0	2.0	6.0	Height of drift is 84 inches.
10- 4-68	-	-	2.4	5.2	
10- 7-68	-	-	2.5	5.2	
10-17-68	-	-	3.0	4.5	
10-21-68	-	91.5	3.0	4.5	
10-24-68	-	90.3	3.8	4.5	
10-25-68	-	90.0	3.8	5.0	
10-28-68	-	88.3	3.0	5.1	Commenced grouting
10-29-68	-	-	2.2	3.6	
10-30-68	-	88.5	2.0	5.0	
10-30-68	-	88.3	1.8	3.6	Performed DOC Grout - Opng. #2
11- 1-68	-	-	2.0	1.8	8:00 a.m.
11- 1-68	-	-	2.0	2.2	2:00 p.m.
11- 4-68	-	89.0	2.0	2.3	
11- 7-68	-	89.5	2.2	2.3	Before regrouting Opng. #1
11- 8-68	-	89.5	2.2	2.3	Finished grouting
11- 9-68	-	-	2.1	2.4	
11-11-68	95.8	90.8	2.2	2.5	
11-20-68	97.8	93.4	2.2	2.9	
11-26-68	99.4	94.8	2.1	2.8	
12- 3-68	99.3	95.0	2.3	2.9	
12-13-68	111.5	96.3	2.9	3.0	
12-18-68	100.5	96.0	2.1	3.0	
12-26-68	103.0	99.0	2.3	3.1	
1- 2-69	105.5	101.3	2.5	3.5	
1- 8-69	105.8	101.1	2.4	3.2	
1-15-69	104.5	99.6	2.3	3.5	
1-24-69	103.3	99.6	2.5	3.4	

TABLE 5 - Cont'd.

WATER MONITORING DATA - MINE NO. 40-016

Date	Fluid Level - (inches)		Flow Rate - (gpm)		Remarks
	Opening #1	Opening #2	Opening #1	Opening #2	
1-30-69	107.0	102.6	3.2	4.2	
2- 4-69	116.3	110.3	3.3	5.0	
2-10-69	117.5	111.3	3.4	4.8	
2-18-69	119.0	114.3	4.3	4.8	
2-24-69	116.5	111.5	3.7	4.5	
3- 3-69	113.3	108.5	3.4	4.6	
3-10-69	110.3	105.8	3.4	4.1	
3-26-69	104.5	100.3	2.8	3.4	
4- 2-69	105.0	100.8	3.1	3.6	
4- 7-69	107.3	102.5	3.1	4.0	
4-14-69	109.5	105.0	3.0	4.0	
4-21-69	108.8	104.3	3.2	3.8	
4-28-69	109.8	105.3	3.8	4.4	
5- 5-69	109.3	104.8	3.6	3.9	
5-12-69	112.0	107.5	3.8	5.2	
5-19-69	114.3	110.0	3.4	4.4	
5-26-69	112.5	108.3	3.6	5.2	
6- 2-69	110.5	106.5	3.2	3.8	
6-10-69	107.5	103.3	3.2	4.0	
6-16-69	105.5	101.5	3.0	3.6	
6-23-69	104.0	99.8	3.0	3.5	
6-30-69	102.5	98.0	2.9	3.4	
7- 7-69	101.0	96.5	2.8	3.4	
7-21-69	101.0	96.8	2.7	3.4	
7-28-69	102.8	98.5	3.2	4.4	
8- 4-69	105.0	100.5	3.4	4.2	
8-11-69	110.0	105.5	3.9	4.6	
8-18-69	114.0	109.8	-	-	

TABLE 5 - Concluded

WATER MONITORING DATA - MINE NO. 40-016

Date	Fluid Level - (inches)		Flow Rate - (gpm)		Remarks
	Opening #1	Opening #2	Opening #1	Opening #2	
8-25-69	114.5	110.0	-	-	
9- 2-69	112.0	107.0	-	-	
9-16-69	107.8	103.3	-	-	Leak through highwall 25 to 30 feet right of Opening No. 1 near base of coal seam
9-29-69	103.5	99.5	2.9	3.4	
10- 6-69	102.5	98.0	3.3	4.2	
10-13-69	101.5	97.5	3.3	4.1	

TABLE 6

WATER ANALYSES - MINE NO. 40-016 - OPENING NO. 1

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
9-20-68	2740	3.2	170	0	1610	149	1690	0.4
10- 4-68	2472	5.2	310	6	1510	50	1508	54.0
10- 7-68	2650	6.1	590	41	1340	192	1690	0
10-17-68	2500	5.4	400	6	1425	142	1664	8.2
10-21-68	2640	5.5	235	20	1590	143	1560	10.2
10-24-68	2520	3.7	130	0	1640	124	1690	27.6
10-25-68	2560	3.4	235	0	1600	144	1664	11.2
10-28-68	2484	3.7	200	0	1600	144	1768	9.2
10-29-68	3180	4.8	210	3	1630	144	1820	23.2
10-30-68	2892	3.6	200	0	1675	152	2028	16.0
10-31-68	2944	3.7	155	0	1720	166	1950	16.0
11- 1-68	2920	3.9	220	0	1680	148	1924	0
11- 7-68	3140	3.1	385	0	1340	163	1638	1.8
11- 8-68	3260	3.1	300	0	1640	154	1768	0
11-11-68	3100	5.8	415	49	1570	178	1820	10.8
11-20-68	2940	5.2	290	13	1580	151	1846	5.8
11-26-68	2900	5.4	180	51	1780	110	1768	6.0
12- 3-68	3020	5.6	130	19	1690	113	1742	4.0
12-13-68	2844	3.6	110	0	1620	101	1664	4.0
12-18-68	2880	4.5	120	0	1650	98	1794	1.0
12-26-68	2840	6.0	100	82	1940	82	1586	0
1- 2-69	2800	6.0	30	112	1730	55	1664	0
1- 8-69	2784	6.1	5	73	1690	67	1612	2.1
1-15-69	2580	5.9	20	70	1750	73	1638	0
1-24-69	2590	5.0	5	11	1260	64	1470	7.5
1-30-69	2800	5.8	110	53	1670	110	1480	72.0
2- 4-69	2680	5.8	48	80	1630	72	1575	0
2-10-69	2560	5.9	123	63	1635	31	1518	<2.0

TABLE 6 - Concluded

WATER ANALYSES - MINE NO. 40-016 - OPENING NO. 1

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/l)	Alka. as $CaCO_3$ (mg/l)	Hardness as $CaCO_3$ (mg/l)	Iron (mg/l)	Sulfate (mg/l)	Aluminum (mg/l)
2-18-69	2400	5.9	363	47	1650	55	1500	0.7
2-24-69	2476	5.9	375	93	1000	55	1320	1.0
3- 3-69	2496	6.5	600	100	1185	108	1505	<2.0
3-10-69	2400	6.4	312	109	1395	73	1575	<2.0
3-17-69	2400	6.7	203	52	1550	99	1400	<4.0
3-27-69	2390	3.6	142	58	1235	69	1470	<5.0
4- 2-69	2380	6.0	107	60	1350	81	1400	<5.0
4- 7-69	2420	6.1	162	83	1360	79	1575	7.0
4-14-69	2314	5.1	48	9	1405	90	1400	0.6
4-21-69	2300	6.0	22	71	1425	66	1295	1.4
4-28-69	2150	5.6	50	37	1755	90	1365	0.4
5- 5-69	2260	5.8	25	55	1360	76	1400	0.7
5-12-69	2240	5.9	30	58	1600	68	2030	2.4
5-19-69	2128	5.6	36	0	1250	74	1350	0
5-26-69	2080	5.9	81	45	1290	61	1400	1.7
6- 2-69	2150	5.7	31	30	955	79	1425	1.1
6-10-69	2146	5.3	51	67	1330	9	1425	1.2
6-16-69	2206	5.6	63	65	1295	80	1450	0.6
6-23-69	2160	5.9	100	40	1560	96	1550	0.1
6-30-69	2160	4.2	105	0	27	113	1250	0
7- 7-69	2178	5.8	130	54	1360	96	1350	0
7-21-69	2308	3.8	185	0	1360	98	1260	1.5
7-28-69	2365	3.3	200	0	305	152	1290	1.6
8- 4-69	2184	3.8	435	0	1355	95	1425	1.0
8-11-69	2190	3.5	325	0	1430	132	1600	4.0

TABLE 7

WATER ANALYSES - MINE NO. 40-016 - OPENING NO. 2

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
9-20-68	2518	4.7	170	0	1570	131	1560	9.2
10- 4-68	2426	5.7	260	27	1520	163	1534	10.0
10- 7-68	2564	6.2	630	57	1250	136	1612	2.2
10-17-68	2436	6.0	280	36	1490	116	1560	6.6
10-21-68	2574	5.8	140	33	1500	110	1638	2.4
10-24-68	2680	3.1	190	0	1575	143	1560	20.2
10-25-68	2460	4.1	115	0	1630	108	1404	17.0
10-28-68	2420	4.9	180	17	1540	120	1664	11.6
10-29-68	3140	5.8	90	40	1670	127	1690	18.0
10-30-68	2892	5.0	125	0	1565	142	1898	0
10-31-68	2844	4.8	140	0	1655	139	1924	18.0
11- 1-68	2860	6.0	90	32	1565	120	1885	0
11- 7-68	2640	5.8	15	41	1660	98	1404	7.0
11- 8-68	2660	5.8	30	42	1700	96	1690	8.0
11-11-68	2822	6.3	25	14	1710	91	1716	2.4
11-20-68	2840	6.7	5	37	1810	82	1820	1.8
11-26-68	2860	6.3	120	148	1700	78	1560	4.4
12- 3-68	2860	6.4	0	131	1800	76	1716	4.4
12-13-68	2684	6.0	30	158	1650	58	1560	7.8
12-18-68	2820	6.0	10	125	1600	78	1664	7.4
12-26-68	2660	6.5	10	201	17[illegible]	49	1508	0
1- 2-69	2600	6.5	20	204	1790	44	1560	0
1- 8-69	2680	6.3	0	152	1800	60	1508	3.8
1-15-69	2440	6.1	10	128	1660	67	1560	0
1-24-69	2500	6.5	410	190	1100	39	1560	<2.0
1-30-69	2700	6.2	70	102	1710	66	1470	<2.0
2- 4-69	2600	6.7	101	173	1430	54	1540	0
2-10-69	2280	6.6	238	191	1220	48	1608	<2.0

TABLE 7 - Concluded

WATER ANALYSES - MINE NO. 40-016 - OPENING NO. 2

Date Sampled	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
2-18-69	2320	6.7	112	181	1205	31	1380	0.5
2-24-69	2334	6.4	75	160	920	49	1380	1.0
3- 3-69	2340	6.9	425	183	1125	81	1295	<2.0
3-10-69	2240	6.5	262	123	1265	50	1365	<2.0
3-17-69	2260	6.7	159	100	1275	73	1435	<8.0
3-27-69	2240	6.2	30	127	1445	60	1400	<5.0
4- 2-69	2332	6.4	17	121	1405	57	1505	<5.0
4- 7-69	2440	6.3	9	160	1530	52	1505	<5.0
4-14-69	2364	6.7	0	140	1570	51	1400	0.4
4-21-69	2280	6.7	155	185	1150	33	1260	1.3
4-28-69	2220	6.5	110	171	1835	29	1120	0.3
5- 5-69	2214	6.4	21	155	1355	43	1260	0.5
5-12-69	2250	6.5	56	156	1320	43	1890	2.1
5-19-69	2120	6.2	10	171	1195	28	1275	0
5-26-69	2008	6.4	93	153	1100	45	1500	1.1
6- 2-69	2110	6.2	27	109	915	55	1375	1.2
6-10-69	2132	5.9	89	127	1245	68	1350	0.8
6-16-69	2112	6.0	0	118	1325	63	1250	0.5
6-23-69	2158	6.0	27	97	1430	66	1400	0.2
6-30-69	2038	6.3	65	81	1235	72	1115	0
7- 7-69	2112	6.1	75	112	1355	59	1225	0
7-21-69	2224	5.9	0	84	1675	70	1260	0
7-28-69	2176	6.3	17	88	1520	64	1250	0.1
8- 4-69	2128	6.3	875	117	1575	62	1200	0.2
8-11-69	2244	6.5	163	177	1490	52	1575	0.6

TABLE 8

PNEUMATIC CONVEYING TEST
3/4-INCH AGGREGATE

Test No.	Aggregate Conveyed (lbs)	Conveying Time (min)	Conveying Rate (lbs/min)	Initial Tank Pressure (psig)	Conveying Line (ft)
1	4820	35.0	138	15	80
2	4780	29.0	165	15	80
3	4605	36.0	128	15	80
4	4550	19.0	239	15	80
5	4270	15.0	285	15	80
6	4310	7.0	616	15	80
7	4220	13.0	325	15	80
8	4220	10.0	422	15	80
9	4200	8.0	525	15	80
10	4200	6.0	700	15	80
11	4180	7.5	557	15	40
12	4200	4.0	1050	15	40
13	4180	7.0	597	15	40
14	4150	6.0	692	15	40
15	4130	5.0	826	15	40
16	4150	4.0	1038	15	40
17	4120	4.5	916	15	60
18	4080	4.0	1020	15	60
19	4000	4.1	976	15	60
20	4000	3.7	1081	15	80
21	4000	5.0	800	15	80
22	4000	4.0	1000	15	80
23	3980	5.5	724	15	80
24	6070	21.5	282	15	80
25	6380	21.0	304	15	80
26	6370	17.0	375	15	80
27	6290	11.0	572	20	80
28	6180	10.5	589	20	80
29	6120	9.4	651	20	80
30	6100	9.3	660	20	80

Note: Air supply for this test was 460 scfm (±5%). The conveying line was a 4-inch ID rubber hose.

TABLE 9

ORGANIC FOAM MATERIALS INVESTIGATION

	Foam A[1]	Foam B[2]
Properties:	Bulk Density - 2 to 2.5 pounds per cubic foot minimum (3.0 pounds per cubic foot minimum possible if ambient temperature is no lower than 60°F) Compressive Strength at Yield Point - 48 psi Tensile Strength - 70 psi Resistance to Dilute Acids - Good Resistance to Dilute Alkali - Exc. Resistance to Degradation by Molds & Fungi - Good	Bulk Density - 1/4 to 2 pounds per cubic foot Compressive Strength - 3 to 5 psi Water Absorption - Very high Resistance to Dilute Acids - Poor Resistance to Biological Attack - Poor Resistance to Water - Poor
Composition:	Polyisocyanate, preheated to 130-150°F Polyol, containing flourocarbon and surfactant - 72°F	Urea-formaldehyde resin - partly polymerized Monobasic ammonium phosphate - catalyst Casein - stabilizer
Cost - Material:	\$2.40 per cubic foot - spray applied at 3.0 pounds per cubic foot	
Cost - Equipment:	\$2,383.00 for Portable Spray Application	\$2,500.00 - \$3,000.00 for Foam Generating Unit

[1] Polyurethane
[2] Urea-formaldehyde

TABLE 10

INORGANIC FOAM TESTS

Components[1]	Formula No. 1360-53-1	Formula No. 1360-53-2	Formula No. 1360-55-1	Formula No. 1360-62-1[2]	Formula No. 1360-64-3[2]
SOLUTION A[3]					
Sodium Silicate - Grade 40	333.5	333.5	50.0	62.5	62.5
Formaldehyde Solution, 37%	102.0	102.0	15.3	25.0	25.0
Water	564.5	564.5	-	62.5	62.5
Howco Suds[4]	-	1.0	-	-	-
Silica Flour	-	-	-	250 g	-
Fly Ash	-	-	-	-	250 g
SOLUTION B[3]					
Sodium Silicate - Grade 40	333.5	333.5	50.0	66.3	66.3
Hydrogen Peroxide Solution, 30%	69.0	69.0	10.4	17.5	17.5
Water	564.5	564.5	5.0	66.3	66.3
Howco Suds[4]	-	1.0	-	-	-
Silica Flour	-	-	-	250 g	-
Fly Ash	-	-	-	-	250 g
Final Foam Volume as % of original fluid volume	230%	350%	153%	280%	203%
Foam Condition[5]	Fragile - crumbles easily.	Very fragile - crumbles very easily.	Much stronger than 1360-53-1.	Good strength - permeable to water.	Good strength.

[1] Components shown as parts by volume except as noted.

[2] 100 cc of each solution mixed.

[3] Equal volumes of Solution A and Solution B mixed in each formula.

[4] Halliburton trade name for a water soluble biodegradable surfactant blend.

[5] Hydrogen gas liberated during generation of foam on all tests.

TABLE 11

INITIAL TEST DATA - SLURRY COMPOSITION

Slurry	Components	Remarks
1	100 parts portland cement[1], 8 parts gypsum, 12 parts bentonite and 109 parts water	Rejected. Not set in 1 hour - medium angle of repose.
2	100 parts cement, 12 parts bentonite and 79 parts water	Rejected. Not set in 3 hours - low angle of repose.
3	100 parts cement, 8 parts gypsum and 46 parts water	Rejected. Not set in 1 hour - low angle of repose.
4	100 parts cement, 8 parts gypsum, 3 parts calcium chloride and 46 parts water	Selected for further consideration. Set in 1 hour - medium angle of repose.
5	20 parts cement, 40 parts fly ash, 40 parts gypsum and 41 parts water	Rejected. Set in 15 minutes. Low angle of repose.
6	75 parts cement, 25 parts gypsum and 38 parts water	Selected. Set in 10 minutes. Medium angle of repose.
7	100 parts cement, 10 parts gypsum, 10 parts bentonite and 3 parts calcium chloride premixed in 92 parts water	Rejected. Angle of repose too low.
8	20 parts cement, 40 parts fly ash, 40 parts gypsum, 4 parts bentonite and 60 parts water	Selected. Set in 10 minutes. Medium angle of repose.

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[1] All cement used in slurry tests was ASTM Type I.

TABLE 11 - Continued

INITIAL TEST DATA - SLURRY COMPOSITION

Slurry	Components	Remarks
9	Same dry materials as Slurry No. 8. Bentonite was prehydrated in 40 parts water. Other materials were mixed with 40 parts water and two slurries blended	Selected. Set in less than 30 minutes - medium angle of repose.
10	Same dry materials as Slurry No. 8. Bentonite and gypsum mixed in 50 parts water. Other materials mixed in 30 parts water and two slurries blended	Rejected. Low angle of repose.
11	Same as Slurry No. 10 except cement and fly ash mixed in 20 parts water	Rejected. Low angle of repose.
12	Same as Slurry No. 10 except bentonite and gypsum mixed in 46 parts water and other materials mixed in 18 parts water before blending slurries	Selected. Set in 30 minutes - medium angle of repose.
13	20 parts cement and 40 parts fly ash mixed in 40 parts water. 40 parts gypsum and 4 parts bentonite mixed in 26 parts diesel oil, then two slurries blended	Selected. Very viscous in less than 1 minute.
14	40 parts cement, 40 parts fly ash, 40 parts gypsum, 4 parts bentonite and 60 parts water. Bentonite prehydrated in 60 parts water	Rejected. Faulty sample. Fly ash would not blend into prehydrated gel slurry

TABLE 11 - Continued

INITIAL TEST DATA - SLURRY COMPOSITION

Slurry	Components	Remarks
15	20 parts cement, 40 parts fly ash, 40 parts gypsum, 2 parts bentonite and 86.5 parts water. Gypsum mixed in 24 parts water and other dry materials mixed in 62.5 parts water, then slurries blended	Rejected. Too thin with low angle of repose.
16	20 parts cement, 40 parts fly ash, 40 parts gypsum and 4 parts bentonite. Bentonite prehydrated in 80 parts water. Cement and fly ash added, then gypsum	Rejected. Did not obtain good distribution of gypsum.
17	Same as Slurry No. 8 except that cement, fly ash and water were mixed together, then blended gypsum and bentonite added quickly to slurry	Rejected. Became too viscous to handle before thoroughly mixed.
18	20 parts cement, 40 parts fly ash, 40 parts gypsum, 2 parts bentonite and 80 parts water. Bentonite prehydrated in 30 parts water, then other materials added separately into slurry	Rejected. 3 hours setting time.
19	20 parts cement, 40 parts fly ash, 40 parts gypsum, 2 parts bentonite, 2.4 parts salt and 80 parts water. Bentonite prehydrated in 30 parts water. Other dry materials mixed in 50 parts water and two slurries blended.	Rejected. Set too slowly.
20	100 parts cement, 10 parts gypsum, 12 parts bentonite and 110 parts water	Rejected. Not set in 1 hour.

TABLE 11 - Concluded

INITIAL TEST DATA - SLURRY COMPOSITION

Slurry	Components	Remarks
21	100 parts cement, 10 parts gypsum, 6 parts bentonite, 3 parts calcium chloride and 110 parts water. Bentonite mixed in 60 parts water. Other dry materials mixed in 50 parts water and alurries blended	Selected. Very viscous but not quick-setting.
22	60 parts cement, 40 parts fly ash, 6 parts bentonite and 100 parts water. Bentonite mixed in 55 parts water. Other dry materials mixed in 45 parts water and slurries blended	Selected. Set in less than 1 hour and very viscous.
23	60 parts cement, 40 parts fly ash, 6 parts bentonite and 110 parts water. Bentonite mixed in 60 parts water. Other dry materials mixed with 50 parts water and slurries blended	Rejected. Not sufficiently viscous.
24	60 parts cement, 40 parts fly ash, 6 parts bentonite, 4 parts calcium chloride and 110 parts water. Mixed same as Slurry No. 23 except that calcium chloride was added to 50 parts water before mixing dry materials or blending slurries	Rejected. Not sufficiently viscous.

TABLE 12

ADDITIONAL TEST DATA - SELECTED SLURRIES

Slurry	Strength, psi, at Various Time Intervals[1]								Material Cost/cu yd
	30 min	1 hr	2 hrs	3 hrs	5 hrs	24 hrs	48 hrs	72 hrs	
4	Firm	Firm	217	400	925	3525	4675	-	$ 31.10
6	Firm	206	240	-	-	1965	2840	3000	32.43
8	160	-	-	140	185	315	-	-	16.80
9	56	46	-	30	38	100	130	170	14.00
12	150[2]	-	-	145	138	275	-	-	15.20
13	Firm	-	105	-	-	170	-	-	24.35
21	Firm	-	-	12	50	300	660	725	18.28
22	Firm	18	35	-	-	261	380	445	18.00

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[1] Compressive strength specimens were placed in a water bath at room temperature until tested. Test was conducted on 2-inch standard test cubes in accordance with Bulletin API RP 10B of the American Petroleum Institute.

[2] Strength shown is for 40 minutes' time.

TABLE 13

QUICK-SETTING SLURRY COMPOSITIONS

Components[1]	Formula No. 1360-61-1	Formula No. 1360-67-1	Formula No. 1360-69-1	Formula No. 1360-72-1	Formula No. 1414-11-1	Formula No. 1414-13-1	Formula No. 1414-17-1	Formula No. 1414-18-1	Formula No. 1414-26-1
CEMENT SLURRY									
Portland Cement	71.43	35.71	32.62	21.74	26.05	21.74	66.67	66.67	66.34
Fly Ash	-	35.71	23.14	-	26.05	-	-	-	-
Silica Flour	-	-	-	15.42	17.30	15.42	-	-	-
Water	28.57	28.58	44.24	29.49	30.60	29.49	33.33	33.33	33.16
Sand	-	-	-	33.35	-	33.35	-	-	-
Howco-Suds[2]									0.50
Slurry Weight, lb/gal	16.23	15.29	13.16	15.21	14.87	15.21	15.27	15.27	15.27
SILICATE SLURRY									
Sodium Silicate[3]	52.63	61.94	33.82	6.55	60.0	7.89	18.69	22.75	22.75
Silica Flour	-	-	35.80	27.74	-	-	62.23	52.77	52.77
Bentonite[4]	-	-	-	2.91	3.0	3.51	0.39	1.73	1.73
Sand	-	-	-	38.78	-	46.71	-	-	-
Water	47.37	38.06	30.38	24.02	37.0	41.89	18.69	22.75	22.75
Slurry Weight, lb/gal	19.82	9.8	12.24	15.17	10.29	12.51	15.00	14.08	14.08
Combined Slurry Wt. lb/gal	13.02	12.54	12.70	15.19	12.58	13.86	15.14	14.68	14.68
Approximate Setting Time, seconds	20	29	35	28	23	24	8	8	8
Estimated Material Cost per cubic yard	$23.77	$17.56	$18.10	$13.10	$17.94	$ 9.47	$26.97	$26.26	$32.68

[1] Components shown as per cent by weight

[2] Halliburton trade name for a water soluble biodegradable surfactant blend

[3] Grade 40

[4] Bentonite prehydrated in water prior to adding other components

TABLE 14

DATA FOR SELECTED SILICATE CEMENT SLURRY

Material Per Yard of Final Slurry[1]

Slurry[2]	Material	Sacks	Gallons	Pounds
1	Portland Cement	12	-	1128
1	Water	-	58	483
2	Bentonite	-	-	50
2	Water	-	75	625
2	Sodium Silicate Grade 40	-	25	292

[1] Cost of final slurry for materials only calculated to be $20.90.
[2] Slurry 1 and Slurry 2 mixed in equal volumes to obtain final slurry.

Compressive Strength - Final Slurry

2 Hours	18 Hours	24 Hours	48 Hours	72 Hours
100 psi	535 psi	990 psi	555 psi[3]	925 psi[3]

[3] Discrepancy thought to be because of difficulty in blending slurries and obtaining sample due to fast set.

TABLE 15

LOW TEMPERATURE TESTS OF BENTONITE-SODIUM SILICATE SLURRY

Test Temperature[1]	Mix Temperature[2]	Ratio of Bentonite to Sodium Silicate	Bentonite (%)	Sodium Bicarbonate (%)	Remarks
30° F	38° F	3 to 1	8	0	Little viscosity increase - not set in one hour
30° F	38° F	3 to 1	8	1.0	Little viscosity increase - not set in one hour
40° F	45° F	3 to 1	8	1.0	Thin - not set in 1 minute - self-supporting[3] in 5 minutes
40° F	45° F	2 to 1	8	0	Viscous in 25 seconds - self-supporting in 1 minute
40° F	45° F	3 to 1	10	0	Not setting in 1 minute - self-supporting in 3 minutes
40° F	45° F	3 to 1	10	1.0	Not setting in 1 minute - self-supporting in 3 minutes.Thinner than slurry above.
40° F	45° F	2 to 1	10	0	Viscous immediately - initial set[4] in 20 seconds
40° F	45° F	2 to 1	10	1.0	Thin for 15 seconds - Set in 20 seconds
40° F	45° F	3 to 1	12	0	Viscous - not self-supporting in 2 minutes
40° F	45° F	3 to 1	12	1.0	Thin for 20 seconds - set in 30 seconds
40° F	45° F	2 to 1	12	0	Viscous - initial set in 30 seconds
50° F	54° F	3 to 1	8	0	Viscous but not set in 10 minutes
50° F	54° F	3 to 1	8	1.0	Thin - initial set in 30 seconds
50° F	54° F	2 to 1	8	0	Viscous - initial set in 30 seconds
50° F	54° F	3 to 1	10	0	Viscous - self-supporting in 2 minutes
50° F	54° F	3 to 1	10	1.0	Thin for 15 seconds - initial set in 20 seconds
50° F	54° F	2 to 1	10	0	Viscous - initial set in 20 seconds

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[1] The bentonite for these tests was mixed in 50° F water at a slow speed on a Dispersator mixer for 20 minutes and then the sodium silicate was added. This slurry and the cement slurry were then cooled to the test temperature indicated before being mixed together at a volumetric ratio of 1 to 1.

[2] Temperature after slurries were mixed together.

[3] Estimated time, by appearance and feel of the material, from time of mixing until it would support itself to permit continued building of a bulkhead.

[4] Time from initial mixing of the 2 slurries until no more stirring could be accomplished with a spatula.

TABLE 16

WATER MONITORING DATA - MINE NO. 62-008 SPECIFIED OPENINGS

Date	S.P. #1[1]	S.P. #4[2]		S.P. #5[3]		S.P. #8[4]	Remarks
	Flow (gpm)	Flow (gpm)	Head (in.)	Flow (gpm)	Head (in.)	Flow (gpm)	
9-20-68	30	-	-	-	-	-	
9-21-68	-	2.8	*	-	*	-	* No bulkheads built yet in any opening.
10- 1-68	48	-	-	-	-	-	
10-17-68	68	-	-	-	-	-	
10-29-68	60	-	-	-	-	-	
11- 8-68	60	-	-	-	-	-	
11-20-68	156	-	-	-	-	-	
11-22-68	-	0	**	-	-	-	** Built silicate cement bulkhead 8 feet back in opening.
11-26-68	98	-	-	-	-	-	
12- 3-68	122	0	-	-	-	-	
2-12-69	-	-	-	15	-	-	
2-13-69	-	-	-	-	†		† Rear silicate cement bulkhead built 28 feet from entrance.
2-15-69	-	-	-	-	††	-	†† Front silicate cement bulkhead built. Grout pipes installed into void between bulkheads
2-27-69	-	-	-	-	-	-	Grouted void between bulkheads-S.P. #5.
2-28-69	-	-	-	-	21.0	-	Head increased from 10.5 inches to 21 inches in 20 hours - opened drain line.

[1] Sample Point #1 - outlet of 48-inch culvert at tipple - Mine No. 62-008.
[2] Sample Point #4 - main Pittsburgh portal designated as Opening No. 4.
[3] Sample Point #5 - utility portal to right of main Pittsburgh portal - Opening No. 5.
[4] Sample Point #8 - fanway opening at right end of other openings - partially collapsed.

TABLE 16 - Continued

WATER MONITORING DATA - MINE NO. 62-008 SPECIFIED OPENINGS

Date	S.P. #1[1]	S.P. #4[2]		S.P. #5[3]		S.P. #8[4]	Remarks
	Flow (gpm)	Flow (gpm)	Head (in.)	Flow (gpm)	Head (in.)	Flow (gpm)	
3-13-69	-	0	-	10.2	-	-	Closed drain lines on both bulkheads.
3-17-69	-	0	-	0	28.5	-	Flowing from S.P. #8 but no measurement taken.
3-26-69	-	0	-	0	24.0	-	S.P. #8 flowing but not measured.
4- 7-69	-	0	-	0	32.5	-	
4-21-69	-	0	-	0	35.0	-	
4-28-69	-	0	42.5	0	38.5	8.6	
5- 5-69	-	0	42.0	0	39.0	7.3	
5-12-69	-	0	43.3	0	41.0	12.5	
5-19-69	-	0	48.3	0	43.8	10.9	
5-26-69	-	0	49.0	0	44.5	8.7	
6- 2-69	-	0	48.8	0	44.8	7.5	
6-16-69	-	0	48.8	0	44.3	3.0	
6-23-69	-	0	48.8	0	44.8	2.3	
6-30-69	-	0	48.5	0	44.5	1.8	
7- 7-69	-	0	48.5	0	44.5	1.8	
7-21-69	-	-	49.5	-	45.8	2.3	
7-28-69	-	0	50.0	0	46.5	2.7	
8- 4-69	-	0	50.8	0	47.0	2.6	
8-11-69	-	0	51.8	0	48.0	3.1	
8-18-69	-	Seep	53.0	0	49.8	3.0	

[1] Sample Point #1 - outlet of 48-inch culvert at tipple - Mine No. 62-008.
[2] Sample Point #4 - main Pittsburgh portal designated as Opening No. 4.
[3] Sample Point #5 - Utility portal to right of main Pittsburgh portal - Opening No. 5.
[4] Sample Point #8 - fanway opening at right end of other openings - partially collapsed.

TABLE 16 - Concluded

WATER MONITORING DATA - MINE NO. 62-008 SPECIFIED OPENINGS

Date	S.P. #1[1]	S.P. #4[2]		S.P. #5[3]		S.P. #8[4]	Remarks
	Flow (gpm)	Flow (gpm)	Head (in.)	Flow (gpm)	Head (in.)	Flow (gpm)	
8-25-69	-	-	48.0	0	50.5	3.0	
9- 2-69	-	-	54.5	-	50.8	3.2	
9-16-69	-	-	54.5	-	51.0	1.9	
9-22-69	-	-	54.3	-	50.8	2.0	
9-29-69	-	-	54.0	-	50.3	1.7	
10- 6-69	-	-	54.0	-	50.3	1.2	
10-13-69	-	-	54.3	-	50.3	1.0	

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[1] Sample Point #1 - outlet of 48-inch culvert at tipple - Mine No. 62-008.
[2] Sample Point #4 - main Pittsburgh portal designated as Opening No. 4.
[3] Sample Point #5 - utility portal to right of main Pittsburgh portal - Opening No. 5.
[4] Sample Point #8 - fanway opening at right end of other openings - partially collapsed.

TABLE 17

WATER ANALYSES - MINE NO. 62-008

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
9-20-68	S.P. No. 1[1]	2854	3.5	180	0	1440	31.7	1560	9.2
10- 1-68	S.P. No. 1	2690	4.5	350	0	1285	18.0	1560	8.6
10-17-68	S.P. No. 1	2620	3.6	365	0	1180	19.2	1560	11.0
10-29-68	S.P. No. 1	2560	4.3	190	0	1330	17.6	1560	7.7
11- 8-68	S.P. No. 1	2350	3.4	260	0	1050	26.4	1352	14.0
11-20-68	S.P. No. 1	2120	3.9	350	0	905	24.6	1443	14.0
11-26-68	S.P. No. 1	2444	3.9	280	0	1120	37.2	1404	21.2
12- 3-68	S.P. No. 1	2426	3.9	250	0	1125	20.4	1378	7.6
9-21-68	S.P. No. 4[2]	5210	3.0	1170	0	1855	246.0	3016	64.0
12-13-68	S.P. No. 5[3]	3060	2.8	2260	0	1235	600.0	4160	88.0

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[1] Sample Point No. 1 - outlet of 48-inch culvert at tipple - Mine No. 62-008.
[2] Sample Point No. 4 - main Pittsburgh portal designated as Opening No. 4.
[3] Sample Point No. 5 - utility portal to right of main Pittsburgh portal.

TABLE 18

SCREEN ANALYSES - GREER LIMESTONE SAMPLES

U.S. Standard Sieve Mesh Size Passed	Retained	Greer #12[1] Per Cent Retained	Greer #12B[1] Per Cent Retained	Greer #13[1] Per Cent Retained
	1/2"[2]	0	-	-
1/2"[2]	3/8"[2]	4.3	0	0
3/8"[2]	4	73.2	38.8	8.7
4	8	21.6	42.0	-
8	16	-	9.8	-
16	50	-	5.8	-
50	100	-	-	71.7
100	Pan	0.9	3.6	19.6

SCREEN ANALYSES - HARROLD LIMESTONE SAMPLES

U.S. Standard Sieve Mesh Size Passed	Retained	Harrold #7[3] Per Cent Retained	Harrold #10[3] Per Cent Retained	Harrold #12[3] Per Cent Retained	Harrold #13[3] Per Cent Retained
	3/4"[2]	0	0	-	-
3/4"[2]	1/2"[2]	85.85	69.90	0	-
1/2"[2]	3/8"[2]	11.83	22.96	-	-
1/2"[2]	3-1/2	-	-	53.0	-
3/8"[2]	4	1.733	5.66	-	4.67
3-1/2	8	-	-	28.0	-
4	8	0.083	0.258	-	6.05
8	20	0.025	0.133	20.0	37.92
20	100	0.275	0.408	0.1	34.18
100	Pan	0.133	0.600	0.6	16.45

[1] Greer Limestone Company, Greer, West Virginia.
[2] Diameter of screen opening.
[3] Paul Harrold Limestone Quarry, Wolf Summit (Clarksburg, W. Va.).

TABLE 19

RETENTION TIME FLOW TEST - HARROLD #12 LIMESTONE[1]

Sample No.	Total Time (hrs)	Retention Time (mins)	pH	Alkalinity as $CaCO_3$ (mg/l)	Total Iron (mg/l)	Dissolved Iron (mg/l)	Ca^{++}	$SO_4^=$
							(mg/l)	
1	1	23.86	7.50	180	10	<1.7	475	1450
2	8	26.90	7.40	160	10	<0.5	575	1450
3	24	23.86	7.45	119	10	<0.5	575	1450
4	48	17.30	7.20	80	5	<0.5	575	1450
5	72	21.20	7.30	100	5	<0.5	575	1400
6	124	66.10	7.60	119	5	<0.5	600	1560
7	158	30.00	7.40	80	40	<0.5	575	1400
8	209	27.50	6.60	39	–	<0.5	575	1450

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[1] Acid mine water used in test had pH of 3.2, acidity of 180 mg/l as $CaCO_3$, total iron of 30 mg/l with 500 mg/l calcium and 1450 mg/l sulfate.

TABLE 20

FLOW TEST DATA - MODIFIED GREER #13 GRADED LIMESTONE[1]

Sample No.	Total Time (hrs)	pH	Alkalinity as $CaCO_3$ (mg/l)	Total Iron (mg/l)	Dissolved Iron (mg/l)	Ca^{++} (mg/l)	$SO_4^=$ (mg/l)
1	1	7.96	160	20	<0.5	500	1550
2	24	7.90	239	60	<0.5	626	1450
3	100	7.86	200	10	<0.5	625	1450
4	125	7.60	160	20	<0.5	626	1400

[1] Acid mine water used in test had pH of 3.2, acidity of 180 mg/l as $CaCO_3$, total iron of 30 mg/l with 500 mg/l calcium and 1450 mg/l sulfate.

TABLE 21

FLOW TEST DATA - MODIFIED GREER #13 LIMESTONE CONTAINING 1% $BaCO_3$[1]

Sample No.	Total Time (hrs)	pH	Alkalinity as $CaCO_3$ (mg/l)	Total Iron (mg/l)	Dissolved Iron (mg/l)	Ca^{++} (mg/l)	$SO_4^=$ (mg/l)
1	1	8.00	239	10	<0.5	625	1450
2	24	7.85	219	10	<0.5	625	1450
3	100	8.00	219	10	<0.5	675	1400
4	125	8.20	259	10	<0.5	675	1400

[1] Acid mine water used in test had pH of 3.2, acidity of 180 mg/l as $CaCO_3$, total iron of 30 mg/l with 500 mg/l calcium and 1450 mg/l sulfate.

TABLE 22

FLUID FLOW DATA
PERMEABLE PLUG TEST NO. 1

	Flow Measurement (gpm)								Standpipe Fluid Levels (Inches)						
	Total		Compartment No.1		Compartment No.2		Compartment No.3		Comp. No. 1		Comp. No. 2		Comp. No. 3		
Date	Mine Flow	Overflow	Input	Discharge	Input	Discharge	Input	Discharge	1-1	1-2	2-1	2-2	3-1	3-2	Comments
12-20-68	28.2	-	9.4	-	9.4	-	9.4	-	-	-	-	-	-	-	Set valves to constant flow
12-23-68	-	-	-	7.5	-	6.0	-	5.0	39	12.0	47	12.3	14	10.5	
12-24-68	28.2	.8	-	8.5	-	7.5	-	5.0	41	11.0	41	8.0	14	10.0	
12-27-68	27.9	13.5	-	3.0	-	5.0	-	6.4	57	12.0	57	15.0	36	18.0	Probed box with metal rod
12-30-68	-	16.2	-	3.0	-	2.5	-	3.8	59	9.5	58	10.5	47	22.5	
12-31-68	-	15.0	1.2	3.8	3.0	2.3	4.2	5.0	60	5.0	60	10.0	52	24.0	#2 Compartment overflowed
1- 2-69	-	20.5	-	2.3	-	1.8	-	2.5	60	12.0	60	10.5	56.5	27.0	#1 Compartment overflowed
1- 3-69	-	24.4	3.3	.1	1.6	.3	3.8	1.0	-	-	-	-	55.0	23.3	Modified #1 input line
1- 6-69	-	22.6	3.3	.9	1.6	.8	3.8	.8	60	9.5	60	7.0	51.5	22.5	Readjusted valves
1- 7-69	-	26.0	.7	.3	.4	.4	1.0	.7	41.5	5.0	52	4.0	48.0	19.0	
1- 8-69	28.2	27.8	.7	.2	.4	.2	1.1	.6	41.5	3.5	54	2.0	42.5	13.0	
1- 9-69	-	27.1	.4	.1	.2	.3	.5	.3	36.3	4.0	53.3	1.5	35.0	8.5	Readjusted valves
1-10-69	-	27.0	.6	.3	.1	.2	.5	.6	53.5	4.0	51.5	3.0	30.5	6.0	
1-13-69	-	27.0	-	.3	-	.1	-	.4	57.5	3.0	63.0	2.0	10.5	5.0	#1 Compartment flooded
1-13-69	-	24.0	.3	-	.3	-	3.6	-	-	-	-	-	-	-	Cleaned and reset valves
1-14-69	-	21.9	.3	.3	.3	.2	5.7	3.0	53.5	3.0	50.5	2.0	49.5	23.0	
1-15-69	-	28.2	.2	.2	.1	.3	5.7	3.7	46.0	3.0	35.0	4.0	43.5	15.0	
1-16-69	-	24.0	.2	.2	.2	.2	3.8	3.5	28.5	3.0	18.0	3.5	34.0	13.0	
1-17-69	-	24.1	.2	.2	.1	.3	3.9	2.6	30.5	3.0	43.0	4.0	30.5	14.0	
1-20-69	-	26.8	0	.2	0	.3	1.4	.9	12.0	3.0	8.0	5.0	19.0	16.0	Cleaned and reset valves
1-21-69	-	26.3	.3	.3	.3	.5	1.4	.8	56.0	4.0	52.5	6.0	36.0	16.0	
1-24-69	28.2	27.6	0	.2	.1	.2	.5	.3	32.5	2.0	45.5	3.0	17.5	9.0	
1-27-69	-	27.6	0	.1	0	.1	.6	.1	51.0	1.5	7.0	1.5	6.0	4.5	Reset valves
1-28-69	-	27.1	.3	.2	.3	.3	.6	.1	57.5	3.5	54.5	5.5	24.0	6.0	
1-29-69	-	31.9	.3	.2	.3	.3	.6	.1	57.0	4.5	55.0	4.0	16.0	9.0	Locally heavy rains
1-30-69	-	-	.1	.2	.1	.1	1.0	.1	54.5	3.5	52.0	4.0	8.0	5.5	
1-31-69	-	32.0	0	.1	.1	.1	.8	.1	40.5	3.0	51.0	3.0	6.0	5.0	Reset #1 valve
2- 3-69	-	48.7	.2	.1	0	.1	.6	.1	40.0	1.0	14.0	.5	4.0	2.0	Locally heavy rains
2- 4-69	-	42.4	.2	.2	.2	.1	.5	.1	57.0	3.0	57.0	3.5	4.5	3.0	
2- 5-69	-	39.6	.2	.5	.2	.6	1.0	.5	56.5	7.0	55.5	5.0	55.5	18.0	
2- 6-69	-	-	-	-	-	-	-	-							Terminated test

TABLE 23

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 1 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
11- 6-68	Mine Flow	3080	3.3	515	0	1165	124.8	1768	25.6
11-26-68	Mine Flow	3080	3.7	580	0	1150	122.4	1820	19.6
12-12-68	Mine Flow	2980	2.9	600	0	1190	120.0	1820	12.6
12-20-68	Comp. #1	2840	5.9	200	44	1780	110.4	1560	0
12-20-68	Comp. #2	2804	6.1	110	67	1390	120.0	1690	1.6
12-20-68	Comp. #3	2840	6.2	70	85	1730	60.0	1430	0
12-24-68	Mine Flow	3040	3.8	580	0	1350	156.0	1820	6.0
12-24-68	Comp. #1	2680	5.8	200	59	1470	108.0	1820	0
12-24-68	Comp. #2	3080	5.9	160	68	1590	120.0	1846	8.0
12-24-68	Comp. #3	2820	6.0	0	107	2270	64.8	1820	0
1- 2-69	Mine Flow	2940	3.8	560	0	1320	129.6	1820	8.6
1- 2-69	Comp. #1	3100	6.1	80	133	1860	117.6	1820	0
1- 2-69	Comp. #2	3200	6.2	0	182	1880	66.0	1820	0
1- 2-69	Comp. #3	3200	6.2	20	164	1990	84.0	1768	0
1- 8-69	Mine Flow	3120	3.2	475	0	1300	121.2	1846	14.4
1- 8-69	Comp. #1	3240	6.3	10	128	1800	62.4	1820	3.8
1- 8-69	Comp. #2	3360	6.3	0	159	1870	48.0	1885	3.2
1- 8-69	Comp. #3	3280	6.2	0	142	1750	62.9	1934	4.8
1-15-69	Mine Flow	2960	3.3	640	0	1200	129.6	1846	15.6
1-15-69	Comp. #1	3060	6.0	60	134	1790	64.8	1885	0.8
1-15-69	Comp. #2	3120	6.2	0	177	1670	40.8	1820	0
1-15-69	Comp. #3	2980	5.7	200	67	1530	98.4	1794	0
1-24-69	Mine Flow	3200	3.2	680	0	1030	25.9	1840	4.7
1-24-69	Comp. #1	3200	6.6	203	160	2380	38.4	2000	8.3
1-24-69	Comp. #2	3300	6.6	189	220	1520	21.6	1845	<2.0
1-24-69	Comp. #3	3200	6.6	168	141	1120	26.4	2000	4.2

TABLE 23 - Concluded

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 1 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
1-30-69	Mine Flow	3010	3.6	510	0	1170	131.9	1800	14.5
1-30-69	Comp. #1	2900	6.6	30	204	1420	17.3	1280	< 2.0
1-30-69	Comp. #2	3325	6.8	60	250	1680	2.4	1750	< 2.0
1-30-69	Comp. #3	3325	6.7	50	252	1650	12.8	1650	2.0
2- 4-69	Mine Flow	3000	3.7	545	0	1390	97.6	1920	13.4
2- 4-69	Comp. #1	3120	6.9	93	254	1350	0.7	2000	0
2- 4-69	Comp. #2	3160	7.0	82	269	1690	1.5	1840	< 2.0
2- 4-69	Comp. #3	3180	7.0	2	267	1870	6.7	1840	< 2.0
2- 6-69	Test Terminated								

TABLE 24

FLUID FLOW DATA - PERMEABLE PLUG TEST NO. 2

Date	Total Input (gpm)	Over-flow (gpm)	COMPARTMENT NO. 1[1]							COMPARTMENT NO. 2[2]							COMPARTMENT NO. 3[3]						
			Input (gpm)	Disch (gpm)	Fluid Level, Inches					Input (gpm)	Disch (gpm)	Fluid Level, Inches					Input (gpm)	Disch (gpm)	Fluid Level, Inches				
					Pt.1	Pt.2	Pt.3	Pt.4	Pt.5			Pt.1	Pt.2	Pt.3	Pt.4	Pt.5			Pt.1	Pt.2	Pt.3	Pt.4	Pt.5
5- 6-69	57.1	36.8	7.4	8.2	-	-	-	-	-	6.4	6.2	-	-	-	-	-	6.5	6.4	-	-	-	-	-
5- 7-69	55.5	32.5	8.1	8.1	-	-	-	-	-	7.7	7.0	-	-	-	-	-	7.2	7.0	-	-	-	-	-
5- 8-69	58.0	39.0	7.0	8.1	12.5	7.3	6.8	4.0	3.3	6.1	7.3	1.0	3.3	4.5	3.3	3.0	5.9	7.0	4.0	3.8	2.8	2.3	2.3
5- 9-69	64.5	41.4	8.4	8.2	16.0	10.3	6.8	4.8	4.0	7.3	6.9	3.5	5.0	6.0	5.0	5.0	7.5	7.0	4.5	4.5	4.8	3.3	3.0
5-12-69	65.1	43.3	8.5	8.5	29.0	13.5	8.5	6.3	3.8	6.2	6.1	7.3	8.3	8.0	7.3	4.3	7.1	6.7	7.8	8.0	8.5	6.5	3.8
5-14-69	57.1	36.1	7.0	7.0	31.5	17.5	10.8	7.3	4.5	7.0	7.0	11.5	12.3	11.5	10.0	6.0	7.0	7.0	13.3	13.5	13.0	10.0	5.8
5-15-69	62.5	41.5	7.0	7.0	-	-	-	-	-	7.0	7.0	-	-	-	-	-	7.0	7.0	-	-	-	-	-
5-23-69	60.0	39.0	7.0	2.7	34.3	33.5	26.5	11.0	6.0	7.0	7.0	17.3	20.0	16.8	18.0	11.5	7.0	7.0	30.0	30.3	24.5	19.3	11.3
5-30-69	60.9	39.9	7.0	2.6	40.8	19.5	14.5	16.0	6.0	7.0	6.9	35.5	36.3	35.5	29.8	16.3	7.0	7.0	30.5	30.5	30.3	27.5	17.5
6- 2-69	64.3	43.3	7.0	1.2	42.3	13.0	12.8	17.5	5.0	7.0	4.7	40.0	40.8	39.5	32.0	18.3	7.0	7.0	32.0	31.3	30.5	30.0	19.0
6- 4-69	63.4	42.4	7.0	0.7	43.8	10.0	8.3	7.8	2.5	7.0	3.4	41.0	42.5	41.8	33.3	19.0	7.0	7.0	33.5	31.3	33.3	30.5	21.0
6- 7-69	64.7	43.7	7.0	0.5	43.8	7.0	4.8	8.5	2.0	7.0	7.0	40.0	42.3	42.5	37.0	20.8	7.0	7.0	36.8	31.8	31.0	30.5	23.3
6-10-69	61.6	40.6	7.0	0.2	43.8	5.5	3.0	2.8	1.8	7.0	7.0	43.5	43.5	43.5	37.3	21.8	7.0	7.0	40.0	34.0	32.0	31.0	24.5
6-16-69	57.1	36.1	7.0	0.1	30.5	3.0	0	0	0	7.0	3.2	42.5	40.5	43.3	37.5	32.0	7.0	5.4	43.0	38.0	38.3	33.8	26.3
6-23-69	49.2	28.2	7.0	0.7	30.0	6.3	4.0	0	0	7.0	0.7	20.5	21.3	20.0	20.8	11.3	7.0	1.1	43.5	39.0	34.0	38.8	14.5
6-30-69	42.5	21.5	7.0	0	18.0	0	0	0	0	7.0	4.4	26.8	24.3	22.5	42.8	25.5	7.0	0.6	32.3	32.3	20.0	15.3	8.0

[1] Compartment covered at 36 inches - filled with #12 limestone.
[2] Compartment open - filled to top with #7 limestone.
[3] Compartment covered at 36 inches - filled with #7 limestone.

TABLE 25

FLUID FLOW DATA - MODIFIED PERMEABLE PLUG TEST NO. 2

			COMPARTMENT NO. 1							COMPARTMENT NO. 2							COMPARTMENT NO. 3								
Date	Mine Flow (gpm)	Input Head (in.)	Input (gpm)	Disch. (gpm)	Fluid Level - Inches Pt.1	Pt.2	Pt.3	Pt.4	Pt.5	Input (gpm)	Disch. (gpm)	Fluid Level - Inches Pt.1	Pt.2	Pt.3	Pt.4	Pt.5	Input Head (in.)	Input (gpm)	Disch. (gpm)	Fluid Level - Inches Pt.1	Pt.2	Pt.3	Pt.4	Pt.5	REMARKS
7- 2-69	-	24	-	4.2	40.5	31.5	15.0	10.3	4.5	-	-						-	-	-						Erected 1' x 2' x 8' high sump extension on Comp. No. 1.
7- 3-69	-	24	-	3.0	37.3	24.5	12.0	8.0	3.0	-	-						-	-	-						
7- 7-69	37.5	24	-	0.7	31.3	7.5	4.8	3.3	1.0	7.0	3.3	40.0	30.3	23.5	23.3	27.0	-	7.0	0.4	27.5	13.3	12.0	8.5	4.8	Perforations plugged in distribution pipe - Comp. No. 2. When opened, water ran over top of rock.
7-21-69	34.7	24	-	0.2	31.5	0	0	0	0	7.0	0	4.5	4.5	2.0	0	0	-	7.0	0.2	10.0	8.0	8.0	4.5	2.3	Perforations plugged in distribution pipe - Comp. No. 2. When opened, water ran over top of rock.
7-28-69	35.7	24	-	0.2	31.0	4.0	0	0	0	7.0	1.6	33.0	14.3	8.0	8.0	31.0	-	7.0	0.2	10.0	11.5	9.5	6.0	3.0	Comp. No. 2 - Water flows over rock to Pt. 5.
7-29-69	-	36	-	-						-	-						-	-	-						Increased head on Comp. No. 1.
8-18-69	49.2	36	-	0.4	30.5	4.0	0	0	0	7.0	0.4	44.5	13.0	13.0	12.3	30.5	-	7.0	0.2	16.0	11.5	11.5	8.5	6.0	Flow into Comp. No. 1 intermittent due to small reservoir capacity.
8-25-69	42.6	36	-	0.2	30.5	0	0	0	0	-	-						-	7.0	0.1	10.5	13.0	13.0	9.5	7.5	Test discontinued for Comp. No. 2.
8-26-69	-	-	-	-													20.0	-	-						Installed lid and pipe riser on Comp. No. 3.
8-28-69	-	36	-	0.2									TEST				27.0	-	12.0						Installed pumping system to supply sump extensions.
9- 2-69	35.1	36	-	0.3	42.5	1.0	0	0	0								29.5	-	7.1	0	43.5	46.0	37.0	21.0	
9-10-69		36	-	-													35.0	-	-						Both compartments now under constant head and flowing.
9-22-69	28.6	24	0.2	0.2	41.5	5.0	0	0	0								38.0	1.9	1.9	-	34.3	39.0	25.0	12.0	
9-30-69	29.7	36	0.5	0.5	53.0	11.0	2	5.5	0				DISCONTINUED				40.0	0.7	0.7	-	22.0	22.5	17.0	5.0	
10-13-69			- Test Discontinued -																- Test Discontinued-						

TABLE 26

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 2 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/l)	Alka. as $CaCO_3$ (mg/l)	Hardness as $CaCO_3$ (mg/l)	Iron (mg/l)	Sulfate (mg/l)	Aluminum (mg/l)
5- 6-69	Mine Flow	2880	3.1	450	0	1015	60.8	1680	10.6
5- 6-69	Comp. #1	2700	5.8	213	32	1400	2.7	1855	1.4
5- 6-69	Comp. #2	2840	3.1	365	0	1050	68.7	1960	6.1
5- 6-69	Comp. #3	2834	3.2	335	0	1150	83.2	1785	7.7
5- 7-69	Mine Flow	2860	3.2	425	0	1075	76.8	1855	10.6
5- 7-69	Comp. #1	2720	5.8	200	28	1305	67.2	1820	2.0
5- 7-69	Comp. #2	2820	3.3	360	0	1125	74.4	1890	2.1
5- 7-69	Comp. #3	2800	3.4	300	0	1085	74.8	1855	8.1
5- 8-69	Mine Flow	2740	3.3	385	0	965	82.8	1820	10.8
5- 8-69	Comp. #1	2640	5.9	335	38	1345	65.2	1925	2.3
5- 8-69	Comp. #2	2720	3.3	360	0	975	67.2	1855	8.9
5- 8-69	Comp. #3	2760	3.3	350	0	1030	64.0	1785	9.1
5- 9-69	Mine Flow	2780	3.1	420	0	500	89.4	1785	11.8
5- 9-69	Comp. #1	2680	5.8	155	35	1275	57.4	1750	2.1
5- 9-69	Comp. #2	2768	3.2	305	0	1115	49.7	2030	9.9
5- 9-69	Comp. #3	2808	3.2	310	0	1210	49.4	1750	9.8
5-10-69	Mine Flow	2800	3.5	450	0	1095	83.8	1820	11.9
5-10-69	Comp. #1	2740	5.9	135	37	1565	60.4	1820	2.2
5-10-69	Comp. #2	2856	3.3	340	0	1190	49.9	1750	9.6
5-10-69	Comp. #3	2820	3.4	305	0	1195	49.3	1855	11.5
5-11-69	Mine Flow	2730	3.4	385	0	1095	84.9	1855	13.3
5-11-69	Comp. #1	2674	6.0	140	49	1375	57.1	1785	2.9
5-11-69	Comp. #2	2780	3.3	320	0	1190	44.4	1785	10.9
5-11-69	Comp. #3	2760	3.3	280	0	1250	48.1	1785	10.2
5-12-69	Mine Flow	2660	3.8	385	0	950	75.0	1750	13.7
5-12-69	Comp. #1	2700	5.9	175	55	1450	56.5	1785	3.0
5-12-69	Comp. #2	2720	3.6	320	0	1125	50.7	1659	10.5
5-12-69	Comp. #3	2680	3.7	275	0	1085	48.9	1575	8.5

TABLE 26 - Continued

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 2 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
5-13-69	Mine Flow	2708	3.6	421	0	829	86.3	1785	13.5
5-13-69	Comp. #1	2668	6.3	140	52	2075	60.6	1785	3.7
5-13-69	Comp. #2	2798	3.6	285	0	2715	44.7	1645	10.5
5-13-69	Comp. #3	2718	4.0	200	0	1275	49.6	1750	8.2
5-14-69	Mine Flow	2714	3.6	400	0	1140	75.2	1715	13.7
5-14-69	Comp. #1	2772	6.7	135	73	1380	49.3	1820	3.7
5-14-69	Comp. #2	2720	4.0	350	0	1150	55.6	1715	8.4
5-14-69	Comp. #3	2650	4.6	251	0	1190	61.8	1645	8.4
5-15-69	Mine Flow	2920	3.1	450	0	1030	84.7	1750	14.5
5-15-69	Comp. #1	2680	5.9	222	32	1180	60.8	1890	0.8
5-15-69	Comp. #2	2860	3.4	320	0	1095	50.6	1820	10.0
5-15-69	Comp. #3	2840	3.2	320	0	1130	47.4	1750	7.4
5-16-69	Mine Flow	2880	3.1	400	0	1050	71.4	1050	12.0
5-16-69	Comp. #1	2700	5.8	170	36	1185	60.7	1715	0.8
5-16-69	Comp. #2	2920	3.2	385	0	1095	48.9	1785	10.0
5-16-69	Comp. #3	2848	3.3	300	0	1145	40.9	1960	7.0
5-19-69	Mine Flow	2730	3.0	441	0	1055	60.7	1820	12.5
5-19-69	Comp. #1	2660	5.6	112	54	1160	39.3	1820	2.0
5-19-69	Comp. #2	2760	3.1	353	0	1125	45.4	1855	9.0
5-19-69	Comp. #3	2700	3.1	205	0	1360	45.9	1799	8.0
5-21-69	Mine Flow	2880	3.0	420	0	1030	46.3	1680	12.0
5-21-69	Comp. #1	2646	6.0	106	46	1415	5.4	1750	7.0
5-21-69	Comp. #2	2806	3.6	325	0	1225	5.7	1470	7.0
5-21-69	Comp. #3	2640	3.5	250	0	1155	43.8	1750	6.0
5-23-69	Mine Flow	2714	3.1	408	0	220	81.4	1680	12.3
5-23-69	Comp. #1	2640	5.7	100	70	1520	59.2	1890	2.5
5-23-69	Comp. #2	2720	2.9	270	0	360	58.5	2070	6.0
5-23-69	Comp. #3	2740	2.9	305	0	1205	54.3	1890	7.9

TABLE 26 - Continued

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 2 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
5-26-69	Mine Flow	2628	2.8	425	0	1070	81.4	1800	14.4
5-26-69	Comp. #1	2720	5.8	15	126	1510	55.8	1830	1.3
5-26-69	Comp. #2	2620	3.2	220	0	1495	69.5	1830	8.7
5-26-69	Comp. #3	2768	2.9	275	0	1455	58.2	1860	10.1
5-28-69	Mine Flow	2750	3.5	450	0	1020	25.2	1800	14.6
5-28-69	Comp. #1	2784	5.9	115	69	1375	51.1	1800	2.7
5-28-69	Comp. #2	2760	3.8	270	0	1275	55.3	1830	8.1
5-28-69	Comp. #3	2880	3.2	365	0	1145	54.1	1680	11.7
5-30-69	Mine Flow	2850	3.1	426	0	1090	85.5	1680	14.3
5-30-69	Comp. #1	2772	6.0	25	70	1490	51.4	1740	2.0
5-30-69	Comp. #2	2864	3.3	230	0	1235	52.6	1680	7.9
5-30-69	Comp. #3	2788	3.2	292	0	1220	61.9	1770	12.7
6- 2-69	Mine Flow	2774	3.4	425	0	1220	95.7	1770	13.1
6- 2-69	Comp. #1	2830	6.2	22	118	1345	44.7	2040	1.3
6- 2-69	Comp. #2	2770	3.5	210	0	1205	62.0	1980	7.6
6- 2-69	Comp. #3	2876	3.1	323	0	1170	66.5	1800	11.4
6- 4-69	Mine Flow	2640	3.7	398	0	1070	74.2	1855	12.4
6- 4-69	Comp. #1	2980	6.2	51	193	1600	117.3	1750	1.3
6- 4-69	Comp. #2	2580	5.8	162	40	1395	59.5	1785	6.4
6- 4-69	Comp. #3	2760	3.7	329	0	1170	67.0	1750	10.7
6-10-69	Mine Flow	2692	3.8	525	0	1065	99.7	1785	14.6
6-10-69	Comp. #1	2928	6.1	134	231	1445	24.3	1855	0.8
6-10-69	Comp. #2	2762	3.4	430	0	1135	92.1	1855	10.3
6-10-69	Comp. #3	2824	3.1	405	0	1140	81.1	1820	11.6
6-16-69	Mine Flow	2644	3.9	465	0	1010	95.4	1820	12.7
6-16-69	Comp. #1	2860	6.4	115	224	1315	4.5	1890	1.4
6-16-69	Comp. #2	2626	5.7	171	22	1320	66.3	1855	7.9
6-16-69	Comp. #3	2666	3.6	430	0	950	80.3	1750	11.1

TABLE 26 - Continued

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 2 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
6-23-69	Mine Flow	2720	3.7	500	0	1015	105.9	1820	13.8
6-23-69	Comp. #1	2760	5.8	23	132	1610	43.4	1785	1.5
6-23-69	Comp. #2	2860	6.1	90	139	1500	49.8	1855	1.3
6-23-69	Comp. #3	2760	6.0	86	174	1520	73.2	1855	4.8
6-30-69	Mine Flow	2770	3.3	503	0	1105	98.7	1505	13.0
6-30-69	Comp. #1	2590	6.0	103	61	138	84.4	1440	0.7
6-30-69	Comp. #2	2710	3.5	425	0	1020	73.8	1610	9.8
6-30-69	Comp. #3	2634	6.4	275	96	1415	73.0	1450	0
7- 7-69	Mine Flow	2572	3.5	525	0	930	103.4	1410	12.2
7- 7-69	Comp. #1	2660	5.9	53	118	1465	70.4	1540	0
7- 7-69	Comp. #2	2540	3.5	375	0	1045	74.9	1530	7.3
7- 7-69	Comp. #3	2654	6.0	20	163	1550	56.3	1295	0
7-21-69	Mine Flow	2558	3.9	425	0	970	88.9	780	13.7
7-21-69	Comp. #1	2684	6.6	16	193	1185	23.6	1380	0
7-21-69	Comp. #2	-	-	-	-	-	-	-	-
7-21-69	Comp. #3	2704	6.6	19	175	1175	34.8	1330	0
7-28-69	Mine Flow	2428	2.9	327	0	1250	95.7	1400	13.0
7-28-69	Comp. #1	2476	6.6	94	190	1370	31.7	1290	0.3
7-28-69	Comp. #2	2492	3.2	280	0	1160	55.6	1260	10.1
7-28-69	Comp. #3	2480	6.9	91	219	1135	18.4	1320	0.5
8- 4-69	Mine Flow	2216	3.4	530	0	890	125.4	1505	13.5
8- 4-69	Comp. #1	2342	6.4	1070	162	1205	41.9	1400	0
8- 4-69	Comp. #2	2108	3.5	310	0	1250	76.5	1435	8.0
8- 4-69	Comp. #3	2424	6.7	1215	209	1370	30.8	1365	0.9
8-11-69	Mine Flow	2282	3.3	495	0	159	88.6	1625	11.6
8-11-69	Comp. #1	2408	6.4	117	199	1020	39.4	1440	0.7
8-11-69	Comp. #2	2256	5.7	370	22	700	58.6	1615	7.0
8-11-69	Comp. #3	2412	6.8	192	246	750	12.1	1410	0.6

TABLE 26 - Concluded

WATER SAMPLE ANALYSES - PERMEABLE PLUG TEST NO. 2 - MINE NO. 40-085

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/1)	Alka. as $CaCO_3$ (mg/1)	Hardness as $CaCO_3$ (mg/1)	Iron (mg/1)	Sulfate (mg/1)	Aluminum (mg/1)
8-18-69	Mine Flow	2320	3.2	395	0	990	96.8	1260	7.9
8-18-69	Comp. #1	2422	6.5	0	184	1270	43.5	1530	0.7
8-18-69	Comp. #2	2354	6.4	0	74	1270	25.9	1560	4.8
8-18-69	Comp. #3	2462	6.9	37	227	1325	32.3	1470	1.2
8-25-69	Mine Flow	2402	6.6	445	0	915	98.2	1500	11.5
8-25-69	Comp. #1	2446	6.5	43	189	1340	41.2	1470	1.5
8-25-69	Comp. #2	1042	5.0	0	126	87	0.5	40	1.7
8-25-69	Comp. #3	2446	4.8	176	204	1230	32.5	1500	2.1
9- 2-69	Mine Flow	2232	3.2	393	0	1040	93.3	1500	13.9
9- 2-69	Comp. #1	2348	6.5	98	192	1380	47.0	1470	0
9- 2-69	Comp. #3	1876	3.3	341	0	1080	81.6	1500	13.7
9-18-69	Mine Flow	2660	3.2	417	0	1160	89.6	1410	12.6
9-18-69	Comp. #1	2740	6.7	26	233	1635	17.8	1295	2.5
9-18-69	Comp. #3	2602	3.6	277	0	1295	70.0	1500	6.4
9-22-69	Mine Flow	2604	2.9	505	0	850	91.2	1440	12.2
9-22-69	Comp. #1	2722	6.8	3	250	1355	18.0	1645	0.2
9-22-69	Comp. #3	2504	6.0	91	51	1350	70.0	1530	1.0
9-30-69	Mine Flow	3032	3.0	293	0	1160	106.0	1520	8.2
9-30-69	Comp. #1	2822	6.7	21	161	1550	-	-	-
9-30-69	Comp. #2	2702	6.3	5	94	1250	87.3	1680	0.6

TABLE 27

FLUID DATA AND WATER ANALYSES
PERMEABLE PLUG TEST NO. 3 - MINE NO. 62-008

Date Sampled	Flow (gpm)	Head (in.)	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/l)	Alka. as $CaCO_3$ (mg/l)	Hardness as $CaCO_3$ (mg/l)	Iron (mg/l)	Sulfate (mg/l)	Aluminum (mg/l)
9-21-68	-	-	4700	4.3	100	0	1490	36.5	1690	3.2
6- 2-69	3.0	-	-	-	-	-	-	-	-	-
6-10-69	3.3	-	4222	3.0	300	0	1445	75.3	2750	5.0
6-16-69	1.8	32.0[1]	4186	6.3	7	255	1595	13.2	2500	0
6-23-69	2.5	33.0	4200	6.5	61	265	1630	7.2	2850	0
6-30-69	2.4	36.3	4160	6.9	58	301	1005	4.2	2550	0
7- 7-69	2.3	39.5	4240	6.4	200	284	1645	6.4	2200	0
7-21-69	2.4	42.3	4202	6.9	11	248	1150	28.7	2200	0
7-28-69	3.4	42.0	3950	6.7	108	234	1830	31.6	2200	0.6
8- 4-69	2.4	42.5	3886	6.8	315	260	1730	26.8	2350	0.1
8-11-69	3.4	44.0	3700	6.6	161	253	1110	24.0	2700	0.4
8-18-69	3.0	44.5	3080	6.7	39	232	1490	45.1	2760	1.2
8-25-69	2.4	46.8	3820	6.7	286	234	1505	33.5	2610	1.7
9- 2-69	4.0	47.3	3724	6.5	222	219	1840	55.1	2340	0
9-16-69	3.2	49.3	4322	6.6	84	182	1250	57.2	2200	2.9
9-22-69	2.2	49.5	4242	6.5	4	200	1205	66.4	2650	0.2
9-29-69	2.2	49.8	4220	6.5	12	181	1935	72.3	2500	-
10- 6-69	3.0	50.0	-	-	-	-	-	-	-	-
10-13-69	2.4	49.8	-	-	-	-	-	-	-	-

[1] Built plug of graded limestone in Opening No. 3 - Mine No. 62-008.

TABLE 28

SCREEN ANALYSES

LIMESTONE USED IN MINE NO. RT5-2[1]

AASHO #67[2] U.S. Standard Sieve Mesh Size Passed	AASHO #67[2] U.S. Standard Sieve Mesh Size Retained	AASHO #67[2] Per Cent Retained	AASHO #8[3] U.S. Standard Sieve Mesh Size Passed	AASHO #8[3] U.S. Standard Sieve Mesh Size Retained	AASHO #8[3] Per Cent Retained
1"[4]	3/4"[4]	7	1/2"[4]	3/8"[4]	11
3/4"[4]	3/8"[4]	78	3/8"[4]	4	76
3/8"[4]	4	97	4	8	97
4	8	99	8	16	99

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[1] Elkins Limestone Co., Inc. - Elkins, West Virginia
[2] Equivalent to West Virginia No. 7 Limestone.
[3] Equivalent to West Virginia No. 12 Limestone.
[4] Size of opening in inches.

TABLE 29

MONITORING DATA

SEAL NO. 1 - MINE NO. RT5-2

Date	Head (ft)	Flow Rate (gpm)	
7-22-69	-	58.5	Bulkhead seal completed 7-20-69 - drain valve still open.
7-22-69	-	0	Drain valve closed at 2:40 p.m.
7-28-69	2.92	-	Small leakage-right-hand side of mine.
7-29-69	3.22	-	Leakage on right-hand side increasing - remedial work necessary.
9-15-69	-	20.1	Remedial work completed by sealing adjacent opening. All drain lines open.
9-17-69	0.81	-	Drain valves closed.
9-18-69	1.20	-	
9-19-69	1.68	-	
9-22-69	2.62	-	
9-25-69	3.25	-	
9-26-69	3.36	-	
9-29-69	3.60	-	
9-30-69	3.62	-	
10- 1-69	3.64	-	
10- 2-69	3.66	-	
10- 3-69	3.70	-	
10- 6-69	3.70	-	
10- 7-69	3.72	-	
10- 8-69	3.72	-	
10- 9-69	3.74	-	
10-13-69	3.78	-	
10-14-69	3.78	-	
10-15-69	3.78	-	

TABLE 30
MONITORING DATA
SEAL NO. 2 - MINE NO. RT5-2

Date	Head (ft)	Flow Rate (gpm)
9-17-69	2.58	1.2
9-18-69	2.92	3.4
9-19-69	3.25	-[1]
9-22-69	3.75	25.0
9-25-69	4.90	21.0
9-26-69	5.00	16.2
9-29-69	5.16	12.1
9-30-69	5.26	9.0
10- 1-69	5.28	9.0
10- 2-69	5.32	16.2
10- 3-69	5.34	16.2
10- 6-69	5.36	16.2
10- 7-69	5.38	9.0
10- 8-69	5.39	4.9
10- 9-69	5.40	3.6
10-13-69	5.44	3.6
10-14-69	5.44	3.6
10-15-69	5.44	3.6

[1] No flow taken due to construction.

TABLE 31

WATER ANALYSES - MINE NO. RT5-2

Date Sampled	Location	Cond. (umhos/cm)	pH	Acidity as $CaCO_3$ (mg/l)	Alka. as $CaCO_3$ (mg/l)	Hardness as $CaCO_3$ (mg/l)	Iron (mg/l)	Sulfate (mg/l)	Aluminum (mg/l)
9-22-69	Opening 2-B[1]	1484	6.2	103	77	895	36.9	780	5
10- 1-69	Opening 1[2]	1400	2.9	696	0	513	104.0	1152	35
10- 1-69	Opening 2-A[3]	1400	2.9	557	0	544	114.0	988	38
10- 1-69	Opening 2-B	1400	6.3	65	0	1279	57.0	940	0
10-13-69	Opening 1	1400	3.1	765	0	657	282.0	916	53
10-13-69	Opening 2-A	1220	3.2	512	0	663	180.0	868	40
10-13-69	Opening 2-B	1160	5.6	0	174	1450	36.0	783	3

[1] Discharge from Opening No. 2 through limestone seal.
[2] Water from behind silicate cement bulkhead in Opening No. 1.
[3] Water from behind limestone seal in Opening No. 2.

BIBLIOGRAPHIC: Halliburton Company, "New Mine Sealing Techniques for Water Pollution Abatement" FWPCA Publication No. 14010 DMO

ABSTRACT: The purpose of this project was to develop and field test new concepts applicable to abatement of acid mine water pollution. Laboratory research determined proper materials, equipment and techniques for constructing mine seals. Two new processes were developed. One involved a technique of placing a plug of graded limestone aggregate in a mine drift to neutralize acid mine water discharge until a seal was effected. In the process precipitation of iron hydroxide gradually closed the pores. A pneumatic conveying technique permitted placement of 550 pounds per minute of aggregate into a mine drift. The second process consisted of remotely constructing a mine seal including rear and front bulkheads of self-supporting, quick-setting sodium silicate cement specifically developed for this application. A filler material of expansive cement was used between the bulkheads to complete the

KEY WORDS:

Research and Development

Acid Mine Water Pollution Abatement

New Mine Sealing Concepts

Self-Supporting Bulkheads

Remote Bulkhead Construction

Permeable Limestone Seals

Acid Neutralization in Situ

seal. Field testing in West Virginia substantiated the feasibility of both processes when two aggregate and two bulkhead type seals were placed in abandoned mines with drainage flows up to 58 gallons per minute. Cost of the seals are reported. This report was submitted in partial fulfillment of Contract No. 14-12-453 between the Federal Water Pollution Control Administration and the Halliburton Company.

Pneumatic conveying Techniques

Mine Sealing Costs

Grouting

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